Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine.

Vol. IX. New York, February, 1911 No. 2.

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York.

Pittsburgh Office, 201 Oliver Building.
Western Office, 1510 Court Place, Denver, Colo.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

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Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, February, 1911.

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The Conquest of the Air by the Chemist.

If we leave aside early and isolated experiments, the endeavor of the chemist to transform the inert elemental nitrogen in atmospheric air into some nitrogen compound of commercial value, has extended on a systematic basis and on a relatively large scale over a little more than the last decade. This issue contains two articles particularly interesting with respect to the Birkeland-Eyde and Badische Company plants in Norway and the new Pauling plant in France for the production of nitrates by arc discharges. The more we learn of the different processes—differing essentially in the type and design of the arc furnace employed—the more it appears that with respect to the yield of grams of fixed nitrogen in form of HNO_3 per kw-hour there is not much of a choice between them; hardly much if any progress has been made in this respect over the unlucky Niagara Falls pioneer plant of Bradley and Lovejoy. Perhaps this is not so very astonishing since in the essential principle all the processes are alike.

But progress has been made in raising the concentration of NO in the gas mixture which leaves the furnace, and very much progress in simplifying the design. Without making any hideous comparisons, the enormous advance made in simplifying the construction is evident; this must find its expression in dollars and cents in the first cost of the plant. How important this is for a new industry will be clear to anybody only slightly acquainted with the ways of capitalists, who are, after all, very human beings like the rest of us.

With respect to the yield of fixed nitrogen per kw-hour we may compare the nitrate process with the calcium cyanamide process, which, though essentially different, is also a method of atmospheric nitrogen fixation. If 1 kw-hour produces ·70 grams HNO_3 it binds 16 grams of nitrogen. On the other hand, in good commercial cyanamide practice 1 kw-hour binds some 70 grams of nitrogen in form of cyanamide. Naturally such a comparison, though interesting, is quite one-sided.

In a recent discussion before the New York Section of the American Electrochemical Society Dr. M. Loeb pointed out that as the formation of NO from air at the temperatures of the arc represents equilibrium, the attempt should be made to remove the NO in some way from the air by absorption in some suitable body so that in order to re-establish equilibrium new NO would be formed, etc. Dr. Whitney pointed out that such absorption might be very difficult at the enormous temperature of the arc. Yet the principle is plausible. In Haber's synthesis of ammonia—another instance of fixation of atmospheric nitrogen—such absorption is carried out and makes the process really practical. But here we work in the cold and absorption is comparatively easy. Clearly, the whole subject is in a state of evolution. It was first vigorously attacked by electrochemists with their inborn insurgent spirit. It is now being taken up by conservative business men. Such a combination must prove invincible in the end.

The Tremendous Magnitude of Western Copper Operations.

During 1910 no metal was the subject of such earnest and careful thought as copper. Overproduction, which had been freely predicted, finally became so great that insistent demands were made upon producers to reach some agreement for curtailment of output. One of the large determining factors was the tremendous development of Western low-grade copper properties, commonly spoken of as the porphyry coppers, which permitted operations on so large a scale that great quantities of copper were available at the remarkably low figures of 7, 8 and 9 cents per pound. The unquestioned ability of these companies to produce largely and cheaply, and the preparations being made by similar companies to enter the list of producers during 1910 and 1911, placed the prospects for the copper market in a bad light. Curtailment was necessary, and was accomplished in some measure, about 10,000,000 pounds per month, although the net result was not as encouraging as had been hoped for. Curtailment was urged as the best temporary method of dealing with a bad situation, for the well informed knew it to be a statistical fact that low prices of themselves would not curtail production and that it would be foolish to continue normal or increased production in the mere hope of a recovery of prices and the probability of causing considerable copper to be sold at a loss.

The development of the Western coppers was an important factor for another reason. On account of the character of the deposits and the nature of the treatment, it was possible to place them on a sound financial basis as investments, making them partake more of the nature of an industrial or manufacturing enterprise with known quantities of raw material and fixed costs of production than was usually possible in a mining venture. In brief, it was possible by calculation to see the end from the beginning and determine the probable life of the business. Furthermore, by combining in one company or group of associated companies the mining, concentrating, smelting and transportation features of the business, the profits of each transaction contributed to the net result.

The Utah Copper Company is the largest present producer of copper concentrate. Originally owning two plants, it later acquired that of the Boston Consolidated and then dismantled the smaller of the original two. The original Utah copper mill at Garfield, Utah, had a daily capacity of 6000 tons of ore averaging 2 per cent copper. Improvements made since then have raised this to 10,000 tons, which is the official figure given in the company's last quarterly report. The Boston Consolidated mill had an original capacity of 3000 tons per day, but after its acquisition by the Utah Copper Company it was wholly remodeled so that its capacity was raised to 6000 tons. This gives the Utah company a total capacity of 16,000 tons of ore per day, with the possibility of forcing the tonnage to a higher figure. Operations of such magnitude become a forcible factor in any field, and when it is considered that the cost of production after smelter deductions are made is from 7½ cents to 8¼ cents per pound of copper, it readily will be understood why a policy of curtailment was inaugurated awaiting the normal recovery of the market.

The Nevada Consolidated Copper Company, operating in White Pine County, Nevada, concentrates its ore in the Steptoe Valley mill and is at present treating about 5000 tons of 2 per cent ore daily. If plans are carried out it will eventually have a daily capacity of 15,000 tons. It is constructed in units of 1500 tons capacity, and 10 such units are embodied in the plans for the complete plant. The cost, as given in the reports of the company, is about 7.9 cents per pound of copper produced, after crediting the cost of copper with the income from the railway and from investments in two associated companies.

The Ray Consolidated Copper Company, Arizona, is one of the porphyry coppers which has not yet come into production, but it gives promise of being a large factor. The assured ore in Ray properties has been brought up to 59,000,000 tons, and the acquisition of the stock of the Gila Copper Company adds about 16,000,000 tons, or a total of more than 75,000,000 tons of 2 per cent copper ore. The mill under construction is planned to have a capacity of at least 8000 tons of ore per day, which should give an annual smelter output of 80,000,000 lb. of refined copper. This present estimate may be increased as development proceeds, for in the company's first report it was stated that the mill would have a daily capacity of 5000 tons, and this figure has now been exceeded. The estimate of cost is not in excess of 9 cents per pound of copper.

The Chino Copper Company will operate in New Mexico and is expected to commence production early this year. Its developed tonnage is 19,000,000 tons of 2.59 per cent copper ore, which is exceptionally high grade for deposits of this character. The concentrating plant is designed to treat 3000 tons per day, but, as has been the case with other mills of this character, the estimate probably will be exceeded. On this basis, however, the company will add from 35,000,000 to 40,000,000 lb. of copper to the 1911 production at a cost not over 9 cents.

The newest copper prospect of the first magnitude is the Inspiration, near Globe, Ariz. The ore is a disseminated deposit about 180 ft. thick. The tonnage which has been proved is 19,000,000 tons of 2 per cent copper ore, and this on an area of only 29.9 acres of land. An experimental mill of 75 tons daily capacity is practically completed, and from the results therein obtained will be planned a mill of 5000 tons daily capacity. Undoubtedly the Inspiration will join the goodly company of low-cost producers mentioned above.

The advent of all of these Western companies into the field under full production will be a factor to be reckoned with. It was pointed out a year ago that their output would be taken care of in the normal course of trade if a policy of curtailment were inaugurated to materially reduce visible stocks before the full flood of Western copper began to find its way into the market, but even now signs are not wanting that a copper convention must be entered into if the integrity and prosperity of the copper industry are to be maintained.

Double-Stage Crushing.

In double-stage crushing as practised on the Rand the stamp has been held in higher esteem than the tube mill as a mechanically efficient machine. The supporters of the tube mill have pointed out, however, that in usual practice the stamp is favored by receiving material which does not require reduction in the first stage, or rather, which is already reduced, while the tube mill is discriminated against by charging it with all the material requiring second-stage reduction.

In this connection recent experiments in South Africa by E. H. Johnson have shown a remarkable performance in double-stage crushing. By first separating the fine ore from the stamp feed and delivering it direct to the tube mill he has obtained from 10 stamps and one tube mill a duty of 26.1 tons per stamp in 24 hours, and a final pulp containing 0.5 per cent remaining on a 90-mesh screen.

Compared with single-stage crushing by stamps and fine screens, the above combination showed an increase of 32 per cent in horse-power efficiency and 31 per cent in grinding efficiency. This indicates possibilities for the tube mill which have not yet been fully appreciated, and which will tend to place double-stage crushing on a firmer basis than ever. As a fine comminuter the tube mill is practically sure of its position in any system of stage reduction, regardless of the machines which may be used in connection with it. Differences of opinion exist regarding the value of stamps as coarse crushers, and some metallurgists believe that rolls and chilean mills would serve the purpose better, but in regard to the tube mill opinion is hardly divided.

Some Handicaps to Iron Exports.

Considerable satisfaction is felt over the fact that our exports of iron and steel products in 1910 made a new record for tonnage by fully 15 per cent, the total of scrap, pig iron, unfinished steel, finished rolled steel, wire products, pipe, etc., amounting to about 1,540,000 gross tons, against a previous record of 1,325,740 tons, made in 1906. Of course, every step forward is a gain, but our export movement is still very small compared with our production. Much of the material exported is well advanced in stage manufacture, the declared value averaging about $40 per gross ton, and the tonnage of pig iron represented in our total exports is considerably in excess of 1,540,000 tons, perhaps 1,700,000 tons, but that is only 6 per cent of our total production in 1910.

Now, 6 per cent is a pretty low figure in view of the fact that in the Lake Superior region we have, as is generally admitted, the finest series of iron ore deposits in the world, and that in the Connellsville region we have as fine coking coal as is available anywhere, and elsewhere we have coal which in the by-product oven makes an entirely satisfactory coke. Copper, with which Nature has also bountifully supplied this country, we export very largely, as contrasted with the meager 6 per cent which obtains in iron.

The divergence in the movement of iron and copper suggests that our iron exports are not what they might be under better domestic conditions, and a glance at the world movements in iron confirms the suggestion. For we observe England making about 10,000,000 tons of pig iron a year and Germany 12,000,000 to 13,000,000 tons, and each exporting about 4,000,000 tons of iron and steel products, or from six to seven times as large a percentage of the production as we export of ours. These countries labor under certain disadvantages. England brings much of her ore from over the sea, her own ores being relatively lean, while Germany's ores are lean and far from tidewater.

It is true that our exports are in a measure restricted by commercial considerations. Our iron and steel export movement is not a "dumping" process but a regular commercial operation. Once our iron and steel exports were strictly of a dumping character, when they took the form chiefly of pig iron, steel billets and sheet bars and rails. In recent years a fairly definite understanding has been built up in the international markets whereby better prices, relative to manufacturing conditions, are obtained than formerly, and our exports cannot suddenly be increased without disturbing the markets, to the disadvantage of ourselves as well as of those sellers whom we should thereby displace. Nevertheless, our exports would have grown more rapidly had we not been subject to certain serious handicaps, the chief of these being the high freight and high labor costs.

There is little doubt that we are at the peak in the curve of labor costs in the United States. It is not reasonable to suppose that the constant increase for a dozen years presages a similar advance for another dozen, and the law of compensation seems even now to be preparing to effect an adjustment. To the iron and steel industry, however, relatively little relief is to be expected, for the movement to shorten hours and provide injury and death reliefs promises a counter influence.

In the matter of reduction in cost of transporting materials some relief is promised as the years pass. The suggestion may sound strange at the moment, when the railroads are so strenuously insisting that they must have higher freight rates in order to maintain their position, but we speak of adjustments as between commodities. A peculiar position prevails as between the railroads and the iron and steel producers. The railroads were long the iron trade's best customers, and in the eighties at least the railroads easily consumed 60 per cent of the product of the iron industry. In the nineties they consumed nearly as large a percentage, and their purchases up to the early part of 1907 were also large, though much smaller, in point of percentage, than in the eighties.

Through the fact of the railroads being the iron trade's best customers and through the movement of iron and steel materials being of a mixed character, it came about that freight rates upon these materials were established at a higher level than would likely be selected were the whole problem to be worked out in the light of present-day conditions. The ton-mile rates paid by the iron and steel industry upon ore, coal, coke, limestone, pig iron, unfinished steel and finished steel average fully as high as the average of all rates paid upon all freight moved over the railroads of the country. This is unnatural. The commodities involved are among the least subject to injury, and are among the most easily handled, of all freight. Enormous tonnages are moved in steady processions, frequently in solid trains involving the minimum expense. Of the value of a ton of finished steel at seaboard ready for export the freight paid in producing it forms a very large part, and there is reason to hope that in course of time this part will be reduced, so that the iron industry will pay no more than its fair share of the expenses of the railroads in conducting their operations.

We do not wish to be understood as intimating that American railroads would suffer by a comparison with those of other countries, for that is not our belief. But we do not think that at present the iron industry is enjoying its full share of the economies effected in American railroading, whereby they are able, as we understand, to stand comparison with English and continental European practice.

Minnesota Meeting of the American Chemical Society.

The forty-third general meeting of the American Chemical Society, held at Minneapolis, Minn., from Dec. 28 to Dec. 31, was attended by 275 members and guests and, like all recent meetings of the Society, it was an echo of the general enthusiasm that pervades its membership.

Owing to the fact that over 175 papers were presented, it was necessary to hold meetings of all of the Divisions and the meetings of the Division of Agricultural and Food Chemistry, Division of Fertilizer Chemistry, Division of Pharmaceutical Chemistry, Division of Industrial Chemists and Chemical Engineers, Division of Physical and Inorganic Chemistry and Division of Organic Chemistry were well attended.

A point especially worthy of note was the success of the Biological Section under the chairmanship of Dr. Carl L. Alsberg, of Washington, D. C. Forty-four papers were presented before this section of a strictly chemical nature bearing upon biochemical problems.

The meeting of the Section of Chemical Education, held at a time when all the industrial chemists could meet with them, proved decidedly interesting and the discussion was general.

On the first day of the meeting the chemists all gathered in general session to listen to eight general addresses as follows:

The Lost Arts of Chemistry, by W. D. Richardson.

The Basis of Industrial Efficiency, by A. D. Little.

Synthetic Metals from Non-metallic Elements, by Herbert N. McCoy.

Progress in Food Chemistry, by H. E. Barnard.

Mechanism of Cell Activity, by Carl L. Alsberg.

Waste Wood and Some of Its By-Products, by Geo. B. Frankforter.

The Formation of Carbohydrates in the Vegetable Kingdom, by Wm. McPherson.

The Efficiency of the College Graduate in the Chemical Industry, by Chas. F. Burgess.

The address of the retiring president, Wilder D. Bancroft, on "A Universal Law" was an exposition of the general application of the theorem of LeChatelier to all kinds of natural phenomena as well as those of a chemical nature.

Possibly the "complimentary smoker" in the rooms of the Commercial Club was as thoroughly enjoyed as any one feature of the meeting, for an unusually witty program had been arranged and original songs bearing upon individual chemists and chemical phenomena were thoroughly enjoyed; also the mementos furnished to the visiting chemists were pleasing to all. This, however, was not the only social function of the meeting, for several receptions and teas were arranged for the ladies and the banquet on Friday night, which was attended by some two hundred members, was also graced by their presence.

On Saturday afternoon a large number were treated to a sight-seeing trip covering both the cities of Minneapolis and St. Paul. A number of the chemists also had the privilege of viewing the wonderful paintings of La Farge in the State Capitol of Minnesota. Excursions were made to the following works: Russell-Miller Milling Company, Pillsbury Flour Mills Company, Washburn-Crosby Company, Consolidated Milling Company, Minnesota Linseed Oil and Paint Company, Archer-Daniels Company, Midland Linseed Oil Company, St. Anthony Falls Water Power Company, Cream of Wheat Company and International Stock Food Company.

The announcement of the election of Professor Alexander Smith as President of the Society and of Dr. Louis Kahlenberg, Dr. Frank K. Cameron, Dr. Geo. B. Frankforter, Dr. E. C. Franklin and Dr. E. G. Love as Councilors-at-Large met with hearty approval and applause.

Prof. M. C. Whitaker was elected editor of the *Journal* of Industrial and Engineering Chemistry and the report of a committee recommending a decided broadening of the policy of that journal was adopted.

The Secretary, Dr. Charles L. Parsons, Durham, N. H., announced a net gain of 550 members for the year, the Society now having a membership of over 5100.

Inventors' Guild.

In the early part of 1910 several men who had done work along the line of invention and who, in developing and patenting their inventions, had come to realize the difficulties and disadvantages under which the inventor labors, instituted a movement for the formation of a society looking toward the betterment of these conditions. The result of this movement was the formation and incorporation in New York City of the Inventors' Guild, the object of which is briefly outlined in the following quotation from the constitution of the society.

"The object of the guild is to advance the application of the useful arts and sciences, to further the interests and secure full acknowledgment and protection for the rights of inventors, to foster social relations among those who have made notable advances in the application of the useful arts and sciences."

Some of the handicaps to which the inventor is subject, other than the proverbial one of never having any money, are the delays in the Patent Office and the ineffectiveness of its work, due to overcrowding and lack of proper facilities in the Patent Office; the expense and tardiness of litigation and the possibility under which a rich corporation, by delaying and prolonging a suit, may increase the expenses to a point which makes such suits prohibitive for a poor inventor; the disadvantage to which the American inventor is subject in foreign patent offices as compared with the liberality of the American Patent Office toward the foreign inventor, and many other conditions militating against the American inventor which should be remedied.

The membership of the Inventors' Guild is limited to 50. The idea of limiting the membership is that with a small society of this sort it is easier to accomplish real results than with a larger organization, hampered as it must be by unwieldiness and red tape. Further, with a small organization each man will feel that he is a working unit and that he will be depended upon to do real work, whereas in a large organization the general feeling is that there will be plenty of other men to do the work, and that lack of assistance from any particular member will make little if any difference. The result is that in the large organization the work, if any, is usually done by even a smaller number of members than that provided for in the Inventors' Guild.

It is proposed to select the membership of the guild carefully, and to this end a member must be formally proposed by a member of the guild, must be personally known to five members of the guild, must pass the membership committee and board of governors, and finally must be voted upon by the whole membership, 4 per cent of the votes cast being sufficient to reject a candidate. The object of such discrimination is to include among the members of the guild men who not only have made inventions, but who have achieved some measure of success in connection therewith, and who will therefore be capable of exerting some influence; and also that no one shall be admitted not congenial to practically the entire membership.

The officers of the Inventors' Guild are as follows: Ralph D. Mershon, president; Chas. W. Hunt, first vice-president; Chas. S. Bradley, second vice-president; Thomas Robins, secretary, and Henry L. Doherty, treasurer.

At the present time the guild has 29 members, as follows: Bion J. Arnold, Dr. L. H. Baekeland, W. H. Blauvelt, Chas. S. Bradley, Alex. E. Brown, Henry L. Doherty, Thomas A. Edison, Carleton Ellis, Stephen D. Field, James Gayley, Edward R. Hewitt, Peter Cooper Hewitt, Chas. W. Hunt, Dr. John F. Kelly, T. S. C. Lowe, Ralph D. Mershon, Ambrose Monell, Dr. Edwin F. Northrup, Prof. G. W. Pierce, Chas. E. Pope, Dr. Michael I. Pupin, Thomas Robins, Dr. F. Schniewind, C. H. Smoot, Prof. Carl Thomas, F. L. O. Wadsworth, Arthur West, Dr. W. E. Winship, B. F. Wood.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our London Correspondent.)

Magnetic Properties of Iron and Its Alloys.

At a meeting of the Institution of Electrical Engineers early in December, Sir ROBERT A. HADFIELD and Prof. B. HOPKINSON presented a paper on "The Magnetic Properties of Iron and Its Alloys in Intense Fields." The materials for the research carried out by the authors were a series of alloys prepared at the Hecla Works, which had already been extensively investigated and many of their physical properties had been ascertained. They included iron-carbon steels, nickel steels and manganese steels in varying proportions.

The most important results obtained were: (1) Every alloy has a different saturation intensity of magnetism, which occurs mostly in a field of 5000 units, but in a few there is a slight increase between 5000 and 25,000. All the materials exhibit the behavior of a mixture of magnetic substances with non-magnetic substances, the permeability of which does not vary materially from unity. The non-magnetic or nearly non-magnetic nickel and manganese alloys show no exception. In none of the alloys examined does the permeability differ from unity by much more than 2 per cent. (2) There is in the series no alloy having a higher specific magnetism than pure iron. (3) The saturation value for pure iron of specific gravity 7.8 is 1675 absolute units, within 1 per cent. (4) In an annealed iron-carbon steel containing a small proportion of other elements the specific magnetism is less than that of pure iron by a percentage equal to six times the percentage of carbon. It is deduced that the magnetism of carbide of iron is about two-thirds of that of pure iron. (5) Quenching an iron-carbon alloy from a high temperature largely reduces this specific magnetism, but the amount of reduction is rather uncertain. (6) The addition of silicon or aluminium to iron reduces the specific magnetism nearly in proportion to the amount added, as if the addition behaved as an inert diluent; but if carbon is present silicon appears to neutralize its action to some extent. (7) In the case of alloys of iron with nickel and with manganese, or with both, no simple relation between their magnetism and composition has been observed. This might be due to the peculiar effects of heat treatment on these alloys. The authors, in conclusion, direct attention to the following general points: Among the alloys there are only one or two whose magnetism is greater than the sum of the magnetism of the separate constituents, a specimen containing 11.4 per cent of nickel being especially noticeable. Secondly, the magnetic properties of every alloy, even those which are almost non-magnetic, are very closely expressed by the equation $B = H + 4\pi I$, where I is the constant designating magnetism, provided that H exceeded 7000 C.G.S. units. Thirdly, comparison with Barrett's results in a field of 40 units, or less, shows that in some cases the magnetism of an alloy may be considerably greater, relative to that of pure iron, in low than in high fields. Barrett found that when the value of H is small several silicon-steels and one aluminium-steel are more magnetic than pure iron; but all these materials are less magnetic than pure iron when saturated. In these cases the coercive force is small. In some alloys with a very high coercive force, such as some of the manganese steels, the magnetism relatively to pure iron is much greater when saturated than in a field of 40—in some cases more than twice greater.

The discussion of the paper was opened by Prof. Silvanus P. Thompson, who said the facts put forward were a strong confirmation of Ewing's theory that the saturation value of intrinsic magnetism was practically a constant. If the research had done nothing beyond establishing the fact that the magnetic behavior of iron is as that of the sum of its constituents, yet a most important point would be established.

Mr. J. T. Morris thought that the research would materially assist in setting up a sound theory for the guidance of experimental work. The influence of temperature was most important. It was interesting to know that this point was now under investigation.

Professor Hopkinson in his reply said he hoped it would be possible to investigate the phases of iron by the magnetic properties of the metal. In the case of annealed carbon-iron alloys he thought it had been shown that the magnetism was a function of the amount of carbon. He did not think that the iron carbide was affected by the presence of iron; and it was clear, both from chemical and microscopical results, that in such alloys they were dealing with mixtures. As to the effect of temperature, he could say that in the case of pure iron a rising temperature lowered the saturation intensity. The reduction increased up to the critical point of 750° C., when the magnetism suddenly disappeared. It seemed probable that temperature affected the molecular constituents; and the question arose whether the reduction of magnetism resulted from the total destruction of the magnetism of certain molecules or whether it was consequent on a gradual change taking place in the whole. He thought the former explanation was more probably correct

Non-Ferrous Alloys for High-Pressure Castings.

At a recent meeting of the Institution of Mechanical Engineers, Prof. H. C. H. CARPENTER and Mr. C. A. EDWARDS contributed an interesting and valuable paper on "The Production of Castings to Withstand High Pressures." The first experiments were made with pure gunmetal containing 85 per cent of copper and 15 per cent of tin, a small quantity of copper phosphide being subsequently added to remove oxygen and leave not more than 0.1 per cent of phosphorus. With this alloy the authors repeatedly reproduced perfectly sound castings which, after being machined, withstood an average pressure of 18 tons before they burst.

Experiments with aluminium bronze followed. The first alloy contained 91 per cent of copper and 9 per cent of aluminium, and the mixture was poured at 1125° C., or about 80° above its solidifying point. Only one unsuccessful casting was obtained out of six. After machining they were tested up to 12 tons to 13 tons, when slight stretching ensued and water came through the valve joint; but by tightenng up the joints the pressure could be raised to 14 tons to 15 tons, when further stretching resulted; but, after again tightening up, the pressure was increased to 18 tons, at which none of the four castings fractured. Consequently, 9 per cent aluminium bronze was capable of withstanding as high pressure as 15 per cent gunmetal; and, moreover, the former had greater ductility.

Up to this point all the castings had been made in iron chills. Further castings were made in green-sand molds, the pouring, as before, being from the bottom. Three castings failed to pass the preliminary test, and the other three were tested up to 10 tons when tightening of the joint was necessary. At 13 tons it stretched again, and, with alternate stretching and tightening of the valve joint, a pressure of 19 tons was attained. The walls were then machined down to varying thicknesses and further experiments carried out which appeared to indicate that if these tests had been carried far enough the cylinders would only have been fractured after the hardening produced by the water pressure had raised the yield stress to the same figure as the ultimate stress.

These results showed that cylinders of aluminium bronze could be cast, both in chills and in sand molds, to withstand a water pressure of 18 tons per square inch, but they had the disadvantage of beginning to stretch at a pressure of 10 tons to 12 tons—the sand casting giving the lower figure. A bronze containing 9.5 per cent of aluminium stood up to 11 tons before bulging, and a 10 per cent bronze was tested up to 12.5 tons before any expansion, while 10.5 per cent bronze stood up to 14 tons before any tightening was necessary. Finally, an 11 per cent aluminium bronze allowed hydraulic pressure to be taken up to 20 tons without any sign of leakage.

In their summary the authors stated that conditions had been

worked out for the production of non-ferrous hydraulic castings capable of withstanding pressures of from 14 tons to 20 tons. These results had been obtained with pure copper-aluminium bronzes containing 9 per cent to 11 per cent aluminium. Their largest experimental castings weighed 36 lb., including gate and riser, and the best proportion of aluminium for these was 9 per cent to 10 per cent. For smaller castings a higher percentage could be employed. The essentials for success were (1) very slow pouring; (2) casting at a temperature not higher than 50° C. to 80° C. above the initial freezing point; (3) the metal must enter the mold at the lowest possible point; (4) green-sand molds must not be too damp, and with close sand the lower part of the mold should be dried; (5) with large castings risers must be used to overcome shrinkage. The average of clean, sound castings was not less than four out of six. Their homogeneity appeared perfectly satisactory, and their corrosion in fresh or in sea water was extremely slight. So far as the authors were aware, no other bronzes possessed this combination of properties.

The discussion was opened by Mr J. Archbutt, of the Midland Railway, who remarked that the paper brought out the importance of attention to small details for the production of good castings. He wished to ask why the authors selected the 15 per cent tin alloy, which was very brittle. Quenching at above 500° C. had a very great influence on the properties of this 15 per cent alloy.

Mr. Bateman said his experience of hydraulic pressures for valves showed that phosphor-bronze would stand three times the working pressure for a long time, while ordinary gunmetal would stand that pressure for only a few weeks.

Mr. Harrison pointed out that commercial engineers had to adhere very rigidly to specification for such customers as the Admiralty. He had found that the metal put into a large casting amounted to 15 times the weight of the finished article.

Mr. C. A. Edwards, in the course of his reply, said it was better to use a big head and ensure a good casting. In the pressure tests they filled the castings with water and pumped up the pressure on top with oil. They had tried quick pouring, and that was why they had so many bad castings at the outset. As to adhering to specifications, if castings would stand higher pressures and be perfectly safe the users should not quibble about other matters.

The Estimation of Phosphoric Acid.

At a recent meeting of the Society of Public Analysts Mr. G. F. Wesley Martin described a new method. The phosphoric acid is first precipitated at ammonium phosphomolybdate, which, after washing, is dissolved in excess of caustic alkali, and the ammonia is then distilled into excess of standard acid and titrated in the usual way. This method has the advantage of fair rapidity, and appears to be sufficiently exact.

Market Report.

December, 1910.

Copper. Opening at £57.5.0 copper remained fairly steady until the close of the month when it showed a downward tendency, closing at £52.15.0.

Tin opened at £169.15.0 and rose smartly to £175 by the 12th, thence dropping to £171.10.0 by the 21st. After this recovered to £175.10.0 (28th).

Lead has been practically fixed at £13.11.3.

Hæmatite opened at 64/6 and appreciated slowly to 66/11½ (28th), the rise being more pronounced after the 9th.

Scotch Pig opened at 55/9 and has kept fairly steady, dropping slightly on the 6th and 13th. Closed at 55/10½.

Cleveland Warrants opened at 49/9, and with the exception of small fluctuations, on the 6th and 13th, have remained steady, closing at 49/10.

Sulphate of ammonia, f.o.b. Liverpool, per ton	£12	17	6
Alum, lump, loose, per ton	5	12	6
Antimony, black sulphide powder, per ton	22	..	..
Borax, British refined crystal, per ton	16	..	..
Sal ammoniac, lump, firsts, delivered U. K.	42	..	..
Copper sulphate, per ton	19	15	..
Camphor, 1-oz. tablets	..	1	7
Caustic soda, ash, 48 per cent, ordinary, per ton	5	10	..
Shellac, standard, T. N., orange spots, per cwt.	4	4	..
Sulphur, recovered, per ton	5	..	..
Carbolic acid, liquid, 97/99 per cent, per gal.	..	1	1
Hydrochloric acid, per cwt.	..	5	..
Creosote, ordinary good liquid	..	..	2
Naphtha, solvent, 90 per cent at 160° C., per gal.	..	..	10
Petroleum, Russian, spot, per gal.	..	..	5½
Copper ore, 10 per cent to 25 per cent unit	9/1½ to 10/1½		
Tin ore, 10 per cent, per ton	£110 to £111		
Mica, small slab, in original cases, per lb.	6d to 1/-		
Litharge, flake, per ton	£15	15	..
Red lead, genuine, per ton	16	5	..
Bleaching powder, 35 per cent, per ton	4	5	..
Platinum, per oz. nominal	8	..	..

India Rubber, Para fine, opened at 6/3½, had dropped by the 7th to 6/1½, and thence to 5/10½ on the 14th and to 5/10 by the 21st, ultimately closing at 5/9 per lb.

Compared with prices for the end of December the following have appreciated:

Antimony has risen per ton	£2	..	..
Copper sulphate	..	5	..
Shellac, per cwt.	..	1	6
Carbolic acid, per gal.	..	..	1½
Creosote, per gal.	..	..	⅛
Naphtha, solvent, per gal.	..	..	½
Petroleum	..	..	¼
Tin ore, per ton	5	..	..
Litharge	..	10	..
Red Lead	..	10	..
Platinum, per oz.	..	6	..

Sulphate of ammonia only is quoted 2/6 cheaper.

London, England.

The Western Metallurgical Field.

Conservation in Metallurgy.

In discussing the efficiency of present metallurgical processes and the possibility of avoiding waste in their operation, Mr. Franklin Guiterman presented some forceful considerations in his annual address as retiring president of the Colorado Scientific Society. By way of introduction the speaker defined the meaning of the word "waste" on account of the "confusion which has arisen in the public mind from the reckless use of the word by superficial magazine writers and that brotherhood of doctrinaires which acclaims itself "conservationists." The waste of a product in whole or in part involves the sacrifice of something which might be profitably produced and which, if saved, could be of use to the community. It therefore follows that while waste is always non-utilization non-use is not necessarily waste.

The metallurgical industry is always at that state of efficiency which limits the further use of raw products of inferior grade, or which makes it inadvisable to carry the process to a higher degree of perfection, even if this be possible. Hence the "waste" of yesterday was due to limiting conditions which may be removed by the progress of to-day, while the improvements of to-morrow are anxiously awaited to reduce to-day's losses. In this sense so-called waste has been merely non-use, and many examples are available to show that the material which was not used years ago is to-day the source of remarkable profits due to improvement in processes of treatment.

An exceptional example of involuntary conservation of an unused product for over 20 years was the retention of some mill tailings from a Leadville mill. Cribbing was erected to prevent the tailing flowing onto neighboring property and into a stream. The product was the waste from concentration of a mixed sulphide ore, of which only the lead and iron content was then useful. The waste assayed Au, 0.04 oz. per ton; Ag, 6 oz. to 7 oz. per ton; Zn, 26 per cent to 27 per cent; Pb, 6 per cent

to 7 per cent; Fe, 20 per cent, and SiO_2, 8 per cent. The zinc ore, blende, was discarded for obvious reasons known to all lead smelters. To allow it to remain in the ore would have been to invite penalties which would have reduced profit to loss. Hence it was eliminated and only retained on the property unwillingly. Twenty years later, owing to the development of electromagnetic separation of pyrite and blende, the 80,000 tons of material in the dump netted the owners over $325,000.

This is probably not typical of anything that will occur 20 years hence with waste or unused products of to-day, for metallurgical progress is more rapid now, and, furthermore, any unused material which has a potential value is experimented with and, generally speaking, is always saved.

Among the unrecovered metals which are either lost in the metallurgical practice of to-day, or recovered only as a contaminating impurity of a more valuable product, are cadmium, bismuth and antimony. Cadmium occurs in almost all Colorado blendes in small quantities. It is a most useful metal for some purposes, but a detriment to its extended use is its high cost of production. Its tendency to rapid oxidation makes its recovery difficult and inefficient, and as it occurs only in small quantities as compared with the zinc, it is practically "wasted."

Bismuth and antimony are recovered in the blast furnace with lead, and contaminate it to the detriment of the price. Bismuth in amounts exceeding 0.4 per cent in lead excludes it from the market for corroding purposes, and the small amount of bismuth could, therefore, be well dispensed with. Here is an example of recovery of a product entailing a financial loss. The same applies to antimony occurring in lead ores, for it increases lead losses in refining and could be dispensed with to commercial advantage.

Another form of so-called waste which has been a fruitful field for those who like to reduce a loss to dollars and cents without regard to its other features is the elimination of sulphur during the smelting of sulphide ores. It would be possible to convert the sulphurous gases into sulphuric acid, but unfortunately there would be no market for this product under governing conditions. The smelters could produce more sulphuric acid in a day than is now made and sold by Western manufacturers in a month. As sulphurous acid is a gas which can be discharged into the air under proper precautions with comparatively little damage to vegetation, and as sulphuric acid is a liquid which must be contained in iron vessels, it is readily seen that the production of the latter would be a matter of large expense rather than of profit, and of serious embarrassment than of congratulation.

Arsenic also is contained in smelting ores, and is now recovered at little or no profit. If this element could be discharged to waste without damage its loss would hardly constitute a failure to observe the laws of conservation. When the time comes that the uses of arsenic warrant an increased production it can readily be supplied at short notice, but until then its recovery is hardly a commercial necessity as far as use is concerned.

Summing up, Mr. Guiterman believes that with respect to gold, silver, lead and copper, and the working costs obtaining in the processes, the limit of efficiency seems to have been reached. There is undoubtedly much room for improvement in the metallurgy of zinc, not only in point of metal recovery but also in the present high cost of smelting operations. But the glaring weaknesses in methods are to be found not so much in the larger as in the smaller enterprises, and it is in respect to the latter that the metallurgical engineer will be called upon to give his counsel and aid.

Progress in Copper Milling and Smelting.

At Needles, Cal., the United States Smelting, Refining & Mining Company has just completed and put in operation a 250-ton concentrator to handle low-grade copper ores. The company also has a smelting plant at Needles and the product of the concentrator will be treated locally.

The initial run of the first unit of the new Miami concentrator in Arizona was to have occurred in January, but delays incident to such a project have postponed operations until the present month so that the operation of the first unit may be coincident with this issue of METALLURGICAL AND CHEMICAL ENGINEERING. The Miami is one of the large low-grade propositions that are being brought into production this year and is modeled on the lines which have proved satisfactory in older mills.

At the Garfield smelter, Utah, reverberatory copper smelting is being improved by running harder silica slags than were formerly considered standard practice. It has been customary to run lower silica slags in reverberatory furnaces than in blast furnaces, as experiment had shown that when silica rose above 38 per cent to 40 per cent in the former the capacity and efficiency of the reverberatory were decreased. Recently, however, reverberatory slags at Garfield have carried as high as 45 per cent silica, and at the Steptoe plant in Nevada as high as 43 per cent, with satisfactory results. This is made possible by exercising greater care in mixing and preparing the charge, and results in the advantage of reducing cost of operation and of fluxing material.

As previously stated in these columns, the California copper smelters have been operating at reduced capacity, or not at all, on account of injunctions obtained by the agriculturists requiring the smelters to abate the smoke and fume nuisance. The U. S. Smelting Company, at Kennett, put a bag house in commission, but it was able to handle the fumes from but two furnaces. Three had been tried but it was found impossible to cool the fumes sufficiently to prevent disintegration of the bags. Later reports, however, indicate that the system has been improved and that three furnaces are now in operation.

The First National Copper Company, owning the smelter at Balaklala, was closed pending the installation of the Cottrell process of electric fume condensation. It was expected that the plant would resume operations in January, as the necessary improvements would be installed by that time. The California metal production has suffered a serious setback as a result of the injunctions obtained by the farmers, and it is doubtful whether, under present market conditions for copper, some of the plants will resume operations with any kind of system of fume recovery.

Montana-Tonopah Mill Operations.

In the eighth annual report of the Montana-Tonopah Mining Company, operating at Tonopah, Nev., some data are given regarding the milling operations. During the year an average of 38.5 stamps out of 40 were dropping continuously. The tonnage milled was 50,245, and the average stamp duty was 3.67 tons per 24 hours. The average value of ore milled was $15.22 per ton (0.206 oz. gold and 21.139 oz. silver). The average value of tailing was $1.43 (0.0146 oz. gold and 2.168 oz. silver). The gross mill extraction was 90.8 per cent, and the cost of milling $3.37 per ton. The bullion produced cost 53.4 cents per ounce, and brought 67.4 cents per ounce.

The Non-Ferrous Metal Market.

With the exception of tin, all the metals suffered a reduction in price during the past month, and while fair business has been reported in all lines, only tin has enjoyed a heavy trade. The prospect for any material change is not considered good.

Copper.—It is generally believed that practically all the business transacted in sales of this metal has been consummated at concessions in price. Even the agencies which have been holding aloof from the market have shown a tendency to enter at current prices. Lake copper is quoted at 12⅝ to 12¾c and electrolytic at at 12.35 and 12.40c.

Lead.—A fair business has been transacted at the following prices: 4.32½ to 4.35c in St. Louis and 4.5c in New York.

Spelter.—Immediate and future deliveries of this metal have combined to make a fair volume of business. St. Louis prices have been about 5.35 to 5.37½c and New York 5.50 to 5.52½c.

Tin.—The continued strength of the foreign market has produced a local conviction that it is due to legitimate causes and buyers are entering the market more liberally. Prices rose somewhat during the month and the ruling price for the month was 39.5 to 40c.

The Iron and Steel Market,

The new year opened with the iron and steel industry running at between 40 and 50 per cent of capacity. Out of a total 33,000,000 tons, the capacity of blast furnaces nominally in blast was about 19,000,000 tons a year, but a capacity of 2,000,000 or 3,000,000 tons was banked to avoid accumulation of pig iron while the attendant steel works were idle. There had been a sharp slowing down during December, on account of seasonable conditions, but there has been relatively little speeding up during January and the gait at the close of January is little if any over 50 per cent of full.

A condition of extreme nervousness prevailed among buyers in December on account of widespread reports that prices of finished steel products might be reduced at the meetings of manufacturers which were expected to be held in connection with the dinner called for the night of Jan. 11. Among producers there was practically no expectation that any change would occur in prices, the dinner being intended rather to encourage adherence to established prices. After Jan. 1 orders began to come in more freely, chiefly in the form of specifications against contracts, and representing requisitions which had been held back to avoid complicating the annual inventories. After the dinner of Jan. 11, with its failure to develop any change in prices, the flow of business experienced further, although slight, improvement.

The total of January bookings in finished steel products promises to show a slight gain over December, and, as January production has been less than December, a gain in unfilled orders on books is likely to be shown. A test of this is furnished by the Steel Corporation's reports. During the whole of 1910 its unfilled orders decreased, but the December decrease of 85,656 tons was the smallest. The total on Dec. 31, 1910, was 2,674,757 tons, the smallest ever reported. A gain is expected to be shown during January of between 60,000 and 120,000 tons.

All expectations of lower prices for finished steel products in January were disappointed, while a surprise, at least to the general trade, occurred in the other direction, since effective Jan. 23 all wire products were advanced $1 per ton, making plain wire 1.55c, base, wire nails $1.75, base, painted barb wire 1.75c and galvanized barb wire 2.05c. The advance was effected by the leading interest announcing its new price, the independents promptly following suit. A conference between a few of the larger interests had previously been held.

The wire advance was made, as such wire advances frequently are, for the purpose of stimulating specifications. The large jobbers had previously been well covered by contracts, permitting specifications to be filed for 60 days, and toward the close of the period it will depend upon the condition of mill order books how much tonnage they will permit to be specified.

The wire advance of $1 a ton follows a decline of $3 a ton during the first seven months of 1910 and successive advances totaling $5 a ton after the great reduction of May 1, 1909, so that the new prices are $3 a ton higher than the lowest prices made after the great market break which started Feb. 19, 1909. Prior to the advance just made the wire mills were running very slack, much less than 50 per cent of capacity, but have been doing better since and will probably be running practically full by the beginning of March.

It seems to be the general opinion that the decline in activity has carried the iron trade well below the normal and that a marked upturn is a certainty, the time only being in doubt, and predictions of a distinct improvement have been made, naming all the dates from Feb. 1 to May 1.

Railroad buying showed a slight improvement in January and there are hopes of further improvement upon the settlement of the freight rate issue by the Interstate Commerce Commission. This is now expected not later than March 15. The Pennsylvania Railroad Company and the rail mills have at last agreed upon specifications for the 150,000 tons of rails for which the railroad offered orders for this year's delivery. This company has ordered about 2000 freight cars and a few other car orders have been placed. No very heavy railroad buying is expected, but a moderate improvement over conditions in the past eight months is to be expected, as many of the roads are short of equipment in one respect or another. There is no prospect of a big rail year. Orders thus far placed indicate a total of between 1,000,000 and 1,500,000 tons for replacement purposes on the steam roads, and with the various other demands the total rail production of the year bids fair to be between 2,500,000 and 3,000,000 tons, not far from the output in 1909 and contrasting with almost 4,000,000 tons produced in 1906, the record year.

Prices of finished steel products are being held quite well. There is occasional shading in one product or another by small interests, but the leading producers, including the Steel Corporation and some eight or ten independents, are holding their prices rigidly, and the attitude of these interests is that they will hold prices irrespective of occasional shading by smaller interests, but will abandon all reserve should any case develop of shading by any large interest. In other words, divergences from the present schedule will hardly prompt an attempt to make a fresh stand upon some level a little lower, but will provoke an open market.

Pig Iron.

Prices have yielded to the extent of about 50 cents in the Philadelphia, Buffalo and Chicago markets, the Pittsburg and Alabama markets being unchanged. There was heavier buying in January than in December, but chiefly by the action of a few consumers who make a practice of buying at this particular time. Merchant pig iron stocks are still accumulating, on the whole, but in some districts stocks are about stationary. Prices at valley furnaces, 90 cents higher delivered Pittsburg, remain as follows: Bessemer, $15; basic, $13.50; No. 2 foundry, $13.75; malleable, $13.50; forge, $13.25. No. 2 foundry is $15.50 at Philadelphia and $14 at Buffalo, No. 2 being $14.25 at Cleveland, $15.50 at Chicago and $11 at Birmingham.

Steel.

Very little business has been done in billets and sheet bars on account of so much tonnage being held over on contracts which should have expired Dec. 31. Prices are very well held on the basis of $23 for billets and $24 for sheet bars, Bessemer or open-hearth, f.o.b. Pittsburg or Youngstown. Forging billets are $28, Pittsburg, with the corresponding price of $30.40, Philadelphia, nominally quoted, while actual sales have been made at $27, Chicago. Rods are $27.50, Pittsburg.

Finished Steel.

Finished steel prices are unchanged, except as to wire products, and are as follows, f.o.b. Pittsburg except where otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, 1.40 cents for tank quality.

Shapes, 1.40 cents for beams and channels, 3-inch to 15-inch inclusive, zees and angles 2 x 3 and larger.

Steel bars, 1.40 cents, base; iron bars, 1.35 to 1.40 cents, Pittsburg; 1.30 cents, Cleveland; 1.30 cents, Chicago.

Wire nails, $1.75, base, per keg; plain wire, 1.55 cents, base; painted barb wire, 1.75 cents; galvanized barbed wire, 2.05 cents.

Black sheets, 28 gage, 2.20 cents; galvanized, 3.20 cents; blue annealed sheets, 10 gage, 1.65 cents; painted corrugated roofing, $1.55 per square; galvanized, $2.75 per square.

Tin plates, $3.60 for 100-lb. cokes.

CORRESPONDENCE

Copper Poisoning.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—We have just had an experience here that seems to warrant the publication of this warning to any one who contemplates working with copper in the electric arc furnace.

We melted 5000 lb. of electrolytic copper scrap—uncontaminated except for a small amount of admixed iron—in a three-phase arc furnace of the ordinary steel furnace type, the experiment lasting some five hours. A few hours after pouring off all of the ten men in the furnace building suffered inconvenience in breathing. For the 24 ensuing hours severe nausea was experienced in each case and a soreness throughout the entire system similar to that of acute grippe.

Since in all other respects the run resembled an ordinary steel run, I attribute the trouble we experienced to copper which was shown to be present in the furnace fumes. The temperature of the copper bath as a whole at no time exceeded 1300° C, so that probably the copper was volatilized only at points directly beneath the electrodes.

From the severity of the attack, coupled with the short time we were exposed to the fumes, and that in a building which was well ventilated, I am of the opinion that very serious results might obtain under less favorable conditions.

I also attach more importance to the above from the fact that in five years of continuous research work on a large scale with furnace processes which involved, among others, tellurium, tellurides, arsenic, arsenides, zinc and sulphide ores, the only physical inconvenience suffered has been an occasional carbon monoxide headache.

C. A. HANSEN.
General Electric Company, Research Laboratory, Schenectady, N. Y.

The Origin of Hydrogen in Blast-Furnace Gas.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In the July issue, 1910, of this journal the authors published an article entitled "A Quantitative Investigation of the Source of Hydrogen in Blast-Furnace Gas." Considerable interest was manifested in the contents of this paper, as it was demonstrated, from the basis of the authors' experiments, that the source of only about half of the hydrogen in the furnace gas could be identified. It will be recalled that the authors suggested that the remainder of the hydrogen in the exit gas might be due to the reaction of hot carbon with combined water in the burden, previously unexpelled by heat.

The above-mentioned article has been reviewed by H. Le Chatelier in the *Revue de Metallurgie* of October, 1910. The main feature of his paper is an explanation of the source of the remaining hydrogen, which he maintains is very simple. He states that the decomposition of water may be readily effected by carbon monoxide in the hot top gases according to the following reaction: $CO + H_2O = CO_2 + H_2$. Le Chatelier states that this reaction begins at 200° C. and occurs to a considerable extent at 300° C., the temperature of the top gas in the authors' experiments, and proceeds to show that at this temperature the entire water content of the gas must be dissociated if the gases are in equilibrium. He admits, however, that in this experiment only half the entire water introduced into the top of the furnace is dissociated, but concludes by stating that the amount would be greater the higher the temperature of the furnace top and the longer the gases remain at this temperature in passing through the mains to the gas-cleaning plant.

The writers have referred to the well-known work of Neuman and Pistor[1] and from their published statements, or from similar work by other authors, fail to confirm the figures in Le Chatelier's article. Neuman and Pistor passed steam and carbon monoxide through a porcelain tube heated to different temperatures and determined the carbon dioxide formed Their results are as follows.

Temp. in Degrees C.	Per Cent CO Converted into CO_2
560	trace
600	2 per cent
900	8 per cent
954	10.5 per cent

These results would show that but little hydrogen would be formed, according to the previously mentioned reaction, below 600° C. But hydrogen may be liberated from water by carbon at a lower temperature. According to Ledebur[2] the decomposition of water by solid carbon begins at 500° C., the product at that temperature being mostly carbon dioxide. From these results it would seem that water would be decomposed by solid carbon rather than by carbon monoxide, since the former reaction sensibly occurs at 500° C. and the latter not until 600° C. is reached.

It would also seem that no reaction could occur in the extreme top of the furnace or in the mains leading from the furnace. The average temperature of the top gas during the experimental work conducted by the authors was only 290° C. The moisture in the gas was not determined in this experiment. Many tests, however, showed the moisture content to vary from 7 per cent to 15 per cent by volume, the hydrogen being at the same time from 4 per cent to 5 per cent.

The recognized complexity of reactions in a blast furnace must cause the solution of the present problem to be difficult. (Even the conditions at the very top of the stock descending into the furnace may be subjected to a sudden change due to slipping.) The writers have been interested in this review by Le Chatelier, but do not think his statement is well substantiated by other published evidence obtainable.

Carnegie Steel Co., Duquesne, Pa. R. J. WYSOR and W. D. BROWN.

Electrode Construction for Furnaces.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—Your January issue, page 42, contains a short article by Dr. Mahlke on a new form of electrode which may involve some features of interest. Permit me to call attention, however, to the prevalent misconception, which I find also exists in this article, that a part of the furnace in which the current density is greater will necessarily be heated to a higher temperature. This is unquestionably an error, even though it has long been supposed to be true. I had called attention to it in some publications on the losses in electrodes.

It was there shown that the stable temperature which any part of the furnace will reach depends not only upon the heat set free in it, but also, and quite decidedly so, on the relative rate at which the heat thus generated internally, or reaching it from other places, will leave that particular part. It was shown that it was quite possible and practicable to use small electrodes with a very high current density, provided they are made so short that the heat will be led off as rapidly as it is formed, and in such a case the temperature will not rise higher than with small current densities.

It would be quite possible, therefore, that the polar enlargements proposed by Dr. Mahlke (and already existing in slightly different forms in other furnaces) could be heated to a higher temperature than the end of the electrode in contact with them, even though the cross-section of the enlargement is much greater than that of the electrode. All that is necessary is that the proportions be such that up to this enlargement all the heat generated in the electrode, and no more and no less, flows out at the cold end. The heat generated in the enlargements then cannot flow out through the electrode, and if it cannot

[1] *Berichte der Deutschen Chemischen Gesellschaft*, XVIII, p. 2894.

[2] "Gasfeuerungen," p. 69.

flow out into any other part as rapidly as it is formed it will accumulate and raise the temperature. Of course, when its temperature gets higher than that of the electrode then there will be a slightly increased heat flow through the electrode, whereby the portion of the enlargement immediately surrounding the hot end of the electrode proper will be slightly chilled by the electrode. It would therefore be possible (whether it would be practicable or not cannot be said off hand) to make this enlarged terminal very great in bulk and actually use it to generate much of the furnace heat, as the heat generated in it could be made to flow into the furnace product lying between these enlarged electrode terminals; the resistance of the product itself might then even be negligibly small. This, therefore, illustrates again the fallacy of our former conception that the temperature must be greater the higher the current density.

Concerning the practicability of his arrangement it must not be forgotten that the electrode generally burns away and must therefore be fed into the furnace; hence anything but a uniform cross-section would in many cases be impracticable. To use the enlargements at the end of electrodes in arc furnaces seems to me to be of theoretical rather than practical interest. It is quite true that the electrode losses in arc furnaces are greatly increased by the fact that the hot end generally has a very much higher temperature than the treatment of the product requires, as, for instance, in iron furnaces; yet to get over this difficulty by the proposed enlargements would probably introduce insurmountable practical difficulties; the furnace, for instance, would have to be opened to replace such an enlargement, not to mention the difficulties in screwing a piece of carbon with a carbon screw to the rounded end of a white hot electrode, from which the stump of the old screw must first be taken out without injuring the thread, all of which seems to be a practical impossibility.

Dr. Mahlke begins his article by saying that this point had been "overlooked" in recent articles on electrode losses. As far as I am concerned I certainly did not overlook it, but in fact even called attention to such enlargements and the possibility of higher temperature in them by referring to the pole plates in the Roechling-Rodenhauser modification of the induction furnace, although I have since then learned that these plates are not counted upon as heat producers.

Philadelphia, Pa. CARL HERING.

Technical Training.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—Co-operation, the slogan for the advancement of the age, hand in hand with specialization, has never before in history been carried on to so great an extent. In the commercial world specialists in each particular line work together toward an ultimate end. The same can be said of the educational world, especially as pertaining to higher education, yet, further, it has been extended to the high schools and, in a moderate degree to the grades. In fact, it has become a recognized principle. When, because of this spirit, such remarkable strides in progress have been made during the last century in each separate field, why, then, arises the question, have they not joined forces? This can practically be accounted for by a certain amount of suspicion with which a member of one fraternity looks upon a member of another. This, of course, has for a basis a slight foundation in that the scientific man's knowledge as to practical things, so thinks the practical man, is about on a par with the farmer's opinion of the city chap and vice versa.

However, the practical man, who has had a practical education and has not been afraid to soil his hands by beginning at the bottom of the ladder, and thus getting a working knowledge of affairs, is the nearest to a combination of the two which we have in this age. Yet, even this man, unless his training has been such as to form in him the student habit, will very soon become so engrossed in the immediate work at hand that he will fail to realize the advance that science is making, and that he is falling behind the procession. No matter how great a success he may be making he may be greater. He must broaden himself. He must know more of the scientific things pertaining not alone to his but as well to allied lines.

The strictly scientific has its place and a very important one in both the advancement of pure and applied science, and where can it be better worked out than in our educational institutions? In fact, the more research, the better, without neglect of the other duties of the instructor, is getting to be a well-recognized principle in the higher educational world. The main reason that more of this is not done is that an inadequate salary occasions the necessity of outside work in order to make both ends meet. Secondly, he is too lazy. Thirdly, he does not have the requisite brain power. When asked why he did not do more in his line I once heard the head of a department in a State university reply, "that it was all worked out and there was nothing more to do." The work for this type of man is digging post holes where there will always be plenty left to dig.

There is rarely found combined in a person the power of original thought, especially when it is logical, and the ability to take a given problem and carry through an investigation to its logical end, whether that end be positive or negative. Generally speaking, there is but one place where it is possible for men to get this training at the present time and this is in the graduate schools of our various educational institutions. It is very seldom that a man who has finished the ordinary four years of any one of our numerous scientific or technical courses in either a college or university has this faculty, at least to any degree.

This may be accounted for, to a certain extent at least, by the fact that the first thing which appeals to the student in engineering and most of the applied sciences is whether or not the particular subject in hand will be of any practical value, i. e., he is looking to his college education as a means to an end and that, in many cases, being the making of a living for himself and later for his family. Now, the courses, for the most part, were planned by men of vast experience who were able to see and know what were the subjects best suited for a man who wishes to prepare himself for a particular profession. Where it is equally possible to appeal to this practical instinct of the student, thus holding his interest in the work at hand without detriment to his training, the easier and further can he be made to progress in a given time.

A plan to bring about this sort of a thing in such a way that it would be possible for the student engaged in obtaining an engineering education to come in contact with and observe men who had obtained the training which he is striving for is by carrying on research of direct practical application. There should be a corps of instructors who are able not only to carry on industrial research themselves and to impart their knowledge to the student, but who can likewise interest men outside who are engaged in technical work to bring their problems to the university or technical school for solution. There they can either avail themselves of the facilities offered by the institution or establish a fellowship for their particular problem similar to those which have been established at the University of Kansas or the University of Pittsburgh.

There will be thus benefited—

(1) The student, who, kept at work by the constant force of inspiration, will learn by association to appreciate and strive to obtain the power of experimentally carrying on a logical line of thought to the desired end. (2) The instructor, who will come in contact with the practical application of scientific ideas, keep pace with the times and thus avoid fossilization. (3) The practical man, by including in his environment men possessed of a fund of scientific knowledge along lines in which he is quite likely to be rusty. Access to a library will do away with useless duplication and he will keep up to date along the line in which he is most interested.

University of Arizona, R. C. BENNER.
Tucson.

Presentation of the Perkin Medal to Dr. Charles Martin Hall.

At the meeting of the New York Section of the American Electrochemical Society, held in the Chemists Club in New York on Jan. 20, 1911, the Perkin medal was formally presented to Dr. Charles M. Hall.

The meeting was well attended, especially by members of the electrotechnical fraternity, who had come to honor Mr. Hall, many of them from outside of New York, Niagara Falls being, of course, well represented. A pretty touch of friendly sentiment and an international flavor were given to the proceedings by the presence and speech of Dr. Paul Héroult, of France.

The Perkin Medal.

The chairman, Dr. CHARLES F. MCKENNA, made the following introductory remarks which we give in abstract:

"The foundation of the Perkin medal was a happy act of scientific benevolence. Perkin's contemporaries in America, wishing to do something which would honor him in life and continue to keep his memory illustrious in science even after he should have passed away, chose to establish this medal and to see to it that in recurring years it should go as a reward to the ones who had struggled, striven and fought and forced nature to yield up further favors to men.

"Sir William Perkin has passed away and now after some further years in which we have witnessed the awarding of this honor we find the following result:

"Each year the achievement of Sir William himself stands out bolder on the pages of chemical history, each year the new award of this medal adds a link to the chain connecting the great work of industrial chemical advancement in America with the earliest and most pronounced advance made in England when aniline yielded up mauve. Each year the appreciation of the honor grows and the Perkin medal with its annual award is coming to be looked upon as the 'Concours' of chemical industry.

"And the thoughtful men who in founding this benevolence, drew up its guarantees and conditions, builded carefully and perhaps better than they knew. Besides leaving the selection of the recipient open to nomination by a large group of American societies interested in chemistry, they set no restriction as to the period in which the discovery may have been made. It is not necessary that the special work shall be that of the preceding year, or of any immediately preceding year. This leads to careful judgment and brings forward for the palm many a struggling chemist who in years past had known the disadvantages of inconspicuousness and the hardships of passing along the dusty road of conflicting interests.

"It happened that the importance of Sir William's own discovery was immediately grasped and that it traveled quickly throughout the world. But this is not always so in our chemical discoveries, and I think it can be said that the full appreciation of the significance of an American college student's experiments in electrolyzing a bath of alumina and kryolite was long in dawning.

"Therefore we must rejoice that the Perkin medal gives the chemical fraternity the opportunity to honor one of our confrères whose modesty has been as singular as his patience has been marked."

CHARLES M. HALL.

Dr. McKenna then introduced the doyen of the chemical profession in America, Dr. Charles F. Chandler, who made the formal presentation speech.

Address of Presentation.

Dr C. F. CHANDLER spoke in part as follows:

"It is my privilege and very pleasant duty as the senior past president of the Society of Chemical Industry to present to Charles Martin Hall, LL.D., the fifth impression of the Perkin medal in recognition of his most valuable work in applied chemistry.

"This work relates particularly to his discoveries and inventions in connection with aluminium. This is a subject which has interested me deeply from my student days. In 1854 I listened to Woehler's account of his discovery of aluminium in 1827, and the following year (1855) I was shown by Rose aluminium which he extracted from cryolite in the spring of that year. In 1869 I had the pleasure of making the acquaintance of Henri St. Claire Deville, who in 1854 and the following years prepared aluminium on a manufacturing scale. Later I had among my pupils at the School of Mines my warm friend Hamilton Y. Castner, who for two years monopolized the manufacture of aluminium by means of his cheap sodium, and finally for the past twenty years I have been very intimate with Dr. Hall, whose electrolytic process at Niagara Falls has produced the metal in such enormous quantities and at such a low price that it has monopolized our market and taken the place for innumerable purposes of other metals formerly employed.

"The name 'aluminium' is derived from alumen, which included alum, in which Margraf in 1754 recognized a peculiar earth which we call alumina and which was supposed a century ago to contain a metal. Sir Humphrey Davy, who in 1807 succeeded in extracting sodium from soda and potassium from potash, and a little later the metals of lime, magnesia, glucina and strontia, endeavored to reduce alumina by heating with the aid of 1000 galvanic plates an iron wire in contact with alumina in an atmosphere of hydrogen. The iron fused, became whiter, and on solution in acid yielded the earth alumina. He evidently produced an alloy of the metals. Nothing further was done till 1827, when Woehler reduced aluminium chloride with potassium and obtained aluminium as a gray, metallic powder, and later in 1845 by acting on aluminium chloride with potassium vapor obtained the metal in tin-white globules.

"In 1854 Henri St. Claire Deville became imbued with the idea of making aluminium a useful metal. He secured the interest of the Emperor Louis Napoleon, and with liberal grants of money was able to work on a factory scale. He employed aluminium chloride and sodium, introduced many improvements in processes for their preparation and application, and manufactured the metal in what for those days seemed very large quantities. Among other improvements he introduced cryolite as a flux. This mineral from Greenland was used experimentally to produce aluminium by the action of sodium in 1855 by both Rose and Percy. Deville's factories continued to produce aluminium on a moderate scale till about 1888, reaching a maximum output of over 5000 lb. per year along in the middle eighties. A few other manufacturers established works. The process of manufacture was, however, so expensive that aluminium was really a precious metal, used chiefly for fancy articles.

The price of metallic aluminium varied before 1886 as follows:

1855—	$90	lb.
1856—	27	"
1857—	22.50	"
1862—	12	"
1886—	12	"

"In 1869 I purchased a little basket in Paris at the aluminium salesroom for $9. It weighed 4 oz.

"In 1886 Hamilton Y. Castner invented a new process for making sodium. He employed caustic soda in place of the carbonate and iron carbide in place of carbon as a reducing agent. As a result he reduced the cost of sodium from $1 per pound to 25 cents, a very important item, as it was never possible to make a pound of aluminium with less than 3 lb. of sodium. By June, 1888, his works at Oldbury, near Birmingham, were in working order, with a daily output of 250 lb., sold at $5 per pound. In 1889 the daily output was 500 lb. at $4 a pound.

"And then the manufacture of aluminium was discontinued because the new electrolytic manufacture from alumina produced the metal at $1 per pound.

"We now come to the achievements of Charles M. Hall, but before I go any further I wish to say that whatever praise I shall bestow on this great inventor I do not in the slightest degree disparage that other great inventor, Paul L. V. Héroult, who worked on the same problem at the same time, across the Atlantic, made the same discovery independently and worked out an analogous process which has played the same important part in Europe that Hall's process has played in America.

"Charles Martin Hall was born in Ohio, Dec. 6, 1863, was educated in Oberlin and graduated there in June, 1885, when 21 years and 6 months of age.

"He studied chemistry while at school, became interested in the aluminium problem while in college and made many experiments in the hope of finding a more economical process for its production. Finally his thoughts turned to electrolysis, and about Oct. 1, 1885, he began to seek a suitable anhydrous solvent for alumina, operative at a practicable temperature, which would bring the alumina into a fluid condition in which it would yield to the electric current.

"Fluorspar and the fluorides of magnesium, sodium, potassium and aluminium were successively tried, but were found to be too infusible and to dissolve little, if any, alumina.

"On Feb. 10, 1886, he tried cryolite, the double fluoride of sodium and aluminium. This fused readily at a very moderate temperature and when he dropped in the white powder of alumina it dissolved and disappeared as does powdered sugar or salt in boiling water. It dissolved to the extent of more than 25 per cent and the solution was clear and limpid.

"He thus made his first great discovery. He proceeded at once to apply the electric current to this solution. His first experiments with clay crucibles were not successful; but on Feb. 23, 1886, he employed a carbon-lined crucible and was successful in obtaining aluminium. He had achieved success in inventing a process for making aluminium directly from alumina by electrolysis under conditions which promised to revolutionize the industry and furnish the metal at so low a price as to enable it to replace copper, tin, lead, zinc and other metals for an endless variety of purposes.

"Hall's age at this time was two months over 22 years and he had discovered and invented what had escaped Woehler, Rose, Deville and many of the other world-renowned chemists who had busied themselves with aluminium over a period of half a century.

"On July 9, 1886, Hall applied for his broad basic patent which was granted April 2, 1889, No. 400,766, with the following claims:

"'1. As an improvement in the art of manufacturing aluminium the herein-described process, which consists in dissolving alumina in a fused bath composed of the fluorides of aluminium and a metal more electro-positive than aluminium, and then passing an electric current through the fused mass substantially as set forth.

"'2. As an improvement in the art of manufacturing aluminium the herein-described process, which consists in dissolving alumina in a fused bath composed of the fluorides of aluminium and sodium and then passing an electric current, by means of a carbonaceous anode, through the fused mass, substantially as set forth.

"'3. As an improvement in the art of manufacturing aluminium, the herein-described process, which consists in dissolving alumina in a fused bath composed of the fluorides of aluminium, sodium and lithium, and then passing an electric current, by means of a carbonaceous anode, through the fused mass, substantially as set forth.'

"He also received four additional patents for modifications of his original process. Hall did not escape the usual experience of the successful inventor. He was obliged to defend in court the originality of his invention. The suit was brought in the United States Circuit Court of the Northern District of Ohio and after an exhaustive discussion of the prior art the patent was sustained. The opinion was written by Judge William Howard Taft, now the President of the United States. He wrote: 'Hall was a pioneer and is entitled to the advantages which that fact gives him in the patent laws.'

"The court declared the patent valid and issued a perpetual injunction against the defendants. For various reasons there was considerable delay in devising suitable means for applying the invention on a manufacturing scale and securing the necessary capital. But at last, in November, 1888, he began work in his factory at Kensington, near Pittsburgh, producing 50 lb. of metal a day. Since that date his progress has been wonderful. He soon erected larger works at Niagara Falls as the first consumer of the electricity supplied by the Niagara Falls Power Company. At the present time the company has three establishments at Niagara Falls, one at Massena in New York on the St. Lawrence and one at Shawinigan Falls in Canada. The total consumption of electricity is 140,000 hp, probably a larger amount than is consumed by any other electrochemical works in the world. The annual output of aluminium is now 40,000,000 lb. The first metal produced at Kensington was sold at $2 per pound; the price now is about 22 cents.

"The early dreams of the young undergraduate have certainly been realized.

"Hall has made many other inventions, several of which have been patented. Six of these are for improved methods for preparing pure alumina, one of which is still employed for preparing this material for use in the works. He has also patented improved methods for making improved carbon anodes for use in the works. There are very many other matters in connection with Hall's work to which I should be glad to call your attention did time permit. I think, however, that I have laid before you a most creditable record of discovery to satisfy you that your committee has been wise and just in awarding to Dr. Hall the gold medal which was established to honor forever the memory of the great chemist, Sir William Henry Perkin, and by association with him to honor our own most distinguished fellow chemists.

"Dr. Hall's achievements certainly entitle him to a place in the front rank of electrochemists and his youthful successes must ever prove an encouragement to other young men who develop a love for this branch of applied science. Personally Dr. Hall is a man of the greatest modesty and lovable character."

In turning to Dr. Hall, Dr. Chandler concluded: "Charles Martin Hall, Doctor of Laws, my dear friend, it gives me the greatest pleasure as the representative of the affiliated chemical and electrochemical societies to place in your hands this beautiful token of appreciation and affection of your fellow chemists."

Personal Reminiscences of Dr. Hall.

Dr. CHARLES M. HALL expressed in a few graceful sentences his thanks for the medal. "I am glad to have this medal on

account of what it represents and on account of the manner in which it has been awarded, and I shall treasure it among my most valued possessions."

Dr. Hall then gave an account of some personal and unpublished facts in connection with his work on aluminium.

"My first knowledge of chemistry was gained as a school boy at Oberlin, Ohio, from reading a book on chemistry which my father studied in college in the forties. I still have the book, published in 1841. It is minus the cover and the title page, so I do not know the author. It may be interesting now to see what this book, published 70 years ago, says about aluminium. It states: 'The metal may be obtained by heating chloride of aluminium with potassium in a covered platinum or porcelain crucible and dissolving out the salt with water. As thus prepared it is a gray powder similar to platinum, but when rubbed in a mortar exhibits distincty metallic luster. Fuses at a higher temperature than cast iron and in this state is a conductor of electricity but a non-conductor when cold.'

"Later I read about DeVille's work in France and found the statement that every clay bank was a mine of aluminium and that the metal was as costly as silver. I soon after this began to think of processes for making aluminium cheaply. I remember that my first experiment was to try to reduce aluminium from clay by means of carbon at a high temperature. I made a mixture of clay with carbon and ignited it in a mixture of charcoal with chlorate of potassium. It is needless to say that no aluminium was produced. I then thought of cheapening the chloride of aluminium then used as the basis for aluminium manufacture and tried to make it by heating chloride of calcium and also magnesium chloride with clay, following the analogy by which iron chloride is produced when common salt is thrown into a porcelain kiln. A little later I worked with pure alumina and tried to find some catalytic agent which would make it possible to reduce alumina with carbon at a high temperature. I tried mixtures of alumina and carbon with barium salts, and also the same mixture with cryolite and carbonate of soda, hoping to get a double reaction by which the final result would be aluminium. I remember buying some metallic sodium and trying to reduce cryolite, but obtained very poor results. I made some aluminium sulphide, but found it very unpromising as a source of aluminium then as it has been ever since.

"On a later occasion I tried to electrolyze a solution of aluminium salt in water, but found nothing but a deposit of alumina on the negative electrode. I did not give a great deal of time to these experiments, as I was a student in college at the time and worked on three or four other attempted inventions. I had studied something of thermochemistry and gradually the idea formed itself in my mind that if I could get a solution of alumina in something which contained no water and in which the solvent was chemically more stable than the alumina, that this would probably give a bath from which aluminium could be easily obtained by electrolysis. The idea remained in my mind and in February, 1886, I began to experiment on this plan. The first thing in which I tried to dissolve alumina for electrolysis was fluorspar, but I found that its fusing point was so high as to be rather discouraging. I next made some magnesium fluoride, but found this also to have a rather high fusing point. I then took some cryolite and found that it melted easily and melted dissolved alumina in large proportion. Then I rigged up a little electric battery—mostly borrowed from my professor of chemistry, Professor Jewett, at Oberlin College, where I had graduated the previous summer of 1885. I melted some cryolite in a clay crucible and dissolved some alumina in it and passed an electric current through it for about two hours by means of carbon electrodes. When I poured out the melted mass I found no aluminium. It then occurred to me that the operation was interfered with by impurities, principally silica dissolved from the clay crucible, so I next made a carbon crucible, inclosed it in a clay crucible and repeated this experiment. This time, after passing the current for a couple of hours, I poured out the material and found a number of small globules of aluminium. I was then quite sure that I had discovered the process that I was after.

"I then undertook to broaden and improve the process and found that I could use, instead of cryolite, other double fluorides, particularly a double fluoride of potassium and aluminium. The most important change, however, which I made at this time was in the material used as an anode. I wanted to get rid of the burning up of the carbon anodes. I tried a platinum anode and found that it seemed to work all right, but it was too expensive. Then I discovered that if I used a fusible bath of a potassium double fluoride with a sodium double fluoride I could use a copper anode, which immediately became coated with a thin film of copper oxide and acted like a permanent platinum anode. This was not a step in advance as I thought it was, as more or less copper gets into the reduced aluminium and the use of a copper anode led me to use very fusible baths which on the whole do not work as well as the less fusible baths. It is probable that this change delayed a successful result for a year or two.

"When worked on a small scale this process with any of the baths I have mentioned, and with either copper or carbon anodes, is not apparently promising. The ampere-hour efficiency is low, sometimes zero, and the bath, whether composed of sodium or potassium salt, becomes filled with a black substance which accumulates and renders the process very difficult. I presume that my friend, Dr. Héroult, whom I have the pleasure of seeing here to-night, who invented the process independently in France about the same time as I did, encountered the same difficulties I did. In spite of the difficulties mentioned, however, I had great faith in the theoretical possibilities of the process and believed that the practical difficulties could be overcome, so I stuck to it from the start.

"On the financial side I had in the course of three years three different sets of partners or backers, of whom two sets became discouraged and gave up. In the summer and fall of 1886 I worked with some people in Boston with whom my brother had made some financial arrangement. The results there obtained were not satisfactory to them and in October, 1886, my Boston friends declined to go further and I returned to my home in Oberlin, Ohio. In December, 1886, I found that a bath composed of a very fusible double fluoride of aluminium and potassium, with copper anodes, worked much better than anything I had before tried. I have here a number of buttons of aluminium made by this method at this time. The larger one was made with current from a galvanic battery on Dec. 7, 1886, and weighs about eight grams. After this I negotiated with the head of one of the large chemical manufacturing companies of the United States with headquarters at Cleveland, but we could not finally agree. I thought then, and have thought since, that this gentleman was the kind of man whom the proverbial inventor meets when he is deprived of the fruits of his labor. He was extremely careful and conservative and was always wanting to take a few days more to think the matter over. He kept me on the string six months and the kind of contract which he finally offered seemed to me too entirely one-sided. The gentleman told a friend of mine afterward that I was no business man, which was no doubt true, but I received what seemed to me much fairer treatment from all with whom I afterward had business dealings and found much more liberal associates and friends.

"The Cowles Electric Smelting & Aluminium Company, who were then making aluminium alloys at Lockport, N. Y., were the second set of people who became interested in my invention. They took an option on it and I spent a year with them, from July, 1887, to July, 1888. They finally threw up their option. The baths which I used at Lockport, at first, worked well for a few days, but after a time they would become less efficient. I finally worked out a system by which the difficulties were overcome. This was by making a bath consisting partly of calcium fluoride or fluorspar and adding 3 per cent or 4 per cent of calcium chloride and using carbon anodes. I reasoned that

chlorine was evolved and burned up the objectionable compound which spoiled the bath. After finally overcoming the difficulties which I have mentioned I made several pounds of aluminium in small crucibles which I showed to Mr. Alfred Cowles and gave him all the facts in relation to the same, but he was not interested.

"I then sent a representative—Mr. Romaine Cole, now dead—to Pittsburgh, who got together the gentlemen who formed the Pittsburg Reduction Company, now the Aluminium Company of America, and we immediately started in the summer of 1888 to build and operate a commercial plant on Smallman Street in Pittsburgh. We had at our disposal about 50 hp in electrical current of 2000 amp. It took a few weeks after starting to get the dimensions of our baths and so forth just right, and then the difficulties which I have referred to disappeared as if by magic. The clogging and spoiling of the bath, which had caused trouble for the last three years, did not occur on a large scale. No calcium chloride was required. It seems that this is a process, unlike a good many others, which works badly on a small scale and well on a large scale. I accounted for this by the fact that on a large scale the electrodes are further separated and there is less circulation between the positive and negative electrodes, which lowers the efficiency and favors the formation of the clogging black compound.

"We also found, as I had predicted nearly three years before and stated in a patent application filed two years before, that on a commercial scale no external heat was required to keep our baths in fusion. This was a great advantage, but I believe that it resulted from a law of nature and not from any invention at the time, as we did not use any excess of current for maintaining fusion, but only the normal current and voltage for electrolytic purposes, and the use of the electric current for fusing and heating in connection with electrolysis was a thing which had been disclosed and published almost a century before.

"The business of manufacturing aluminium has now grown to a great commercial business. Many workers have contributed to it and the credit is to be divided among many. Our financial people, particularly the gentlemen of the Mellon Bank of Pittsburgh, have given indispensable aid in purchasing our stock and in loaning us money most generously. Our patent attorneys and experts, our engineering and chemical staff, sales agents, superintendents, foremen and others have all done their share."

"In the commercial development of the business," Dr. Hall concluded, "the greatest credit is due to the late Captain Alfred E. Hunt and to Mr. Arthur V. Davis."

Personal Reminiscenes of Dr. Héroult.

Dr. Paul Héroult then made the following address:

"There is a verse of Kipling which reads as follows:

> "'Ship me somewhere east of Suez,
> Where the best is like the worst;
> And there ain't no ten commandments,
> And a man can raise a thirst.'

"I was in that particular mood when I made my first acquaintance with aluminium in large quantities. At the time I had a friend who since then became my partner, but for the time being we were both dead broke. We had pawned everything in sight and also other things which were not in sight.

"Finally my partner had a bright idea. He brought back from home a stick of aluminium about 6 in. long which was valued very high in his family as a personal souvenir of Saint Claire Deville.

"As we handed it to the pawnbroker, the latter said: 'What is that, bar silver?' We said: 'Better than that; that is aluminium.' 'Aluminium,' he said, 'what is that?'

"He weighed it in his hand and said: 'Why, is this hollow?' We said: 'No, that is aluminium and it is worth 120 francs per kilo.' After some thought he said: 'Well, I will give you 2 francs for it,' say 40 cents in American money.

"On a hot summer's day it was better than nothing, and we took the money with the firm intention of buying the stick back, which we never did. Maybe that was one of the reasons why, later on, I had to make good and replace it.

"No material change took place in the industry of aluminium for three years after that event, but the actual fact of the development of said industry was preceded by rumors to the effect that something new was coming.

"I can remember there was a shop on the Grand Boulevard in Paris where there was a display of aluminium goods, mostly thimbles, if I remember rightly; besides wedding rings, brooches, statuettes, a few coils of wire and quite some interesting specimens of aluminium bronze in the form of coffee pots, sugar bowls, etc.; none of the cheap stuff which is sold nowadays. I bought once a thimble for my mother out of my savings. I was a good boy in those days. Every one of those articles had on the Paris finish and sold like silverware.

"In the nineties this same shop practised the same prices, although the cost for ingots had come as low as 25 francs per kilo.

"This was a wonderful time for the aluminium makers; they were making money hand over fist, at least on the books. There was no use for any syndicate at that time. Aluminium was being piled up in the store and the stock of finished product reckoned at above-named price. But this did not last long; customers were scarce and finally the bankers got tired of lending money to people who piled up stock; it became necessary to sell.

"Another price was established at 8 francs per kilo which did not prove successful either. Then came the 5 marks which originated with the Swiss Company; finally it seemed to open the bunghole of the cask.

"Now after various vicissitudes the price in the old country looks pretty near settled around 20 cents per pound and will vary probably according to the price of copper.

"Twenty-five years ago the aluminium industry employed about 10 men; the production was 1½ tons a year, the total value of which was $30,000. Nowadays the same industry produces 50,000 tons at a vaiue of $20,000,000 and employs from 15,000 to 20,000 men.

"If we take the average consumption of copper at 900,000 tons a year at a value of 13 cents per pound, the total value of ingot metal will be $234,000,000 a year comparing with the $20,000,000 for aluminium, say one-tenth. But 13 cents is pretty near the rock bottom price for copper. At 20 cents per pound aluminium presents a great advantage for most uses.

"Aluminium is slowly wedging its way in the metal market. It looks probable that in a course of 10 or 15 years the consumption of the new metal will be equal to that of copper and that after the golden age, the stone age, the bronze age and the iron age we will have the aluminium age.

"My friend Hall and myself have been fighting for 15 years, most of the shots going wild on account of the long reach over the Atlantic.

"Since we met, however, we conceived a better opinion of each other and I take great pleasure to-night in extending to my friend Hall my sincere congratulations on the award to him of the Perkin medal."

Concluding Speeches.

Dr. Jos. W. Richards then gave some reminiscences of his friendship of many years with Dr. Hall. He emphasized that Hall had taken at Oberlin College the classical, not the scientific, course. It seems as though a broad foundation such as given by a classical course is a very suitable basis for later technical work.

Dr. Richards then gave a brief review of the various uses of aluminium. First, in the steel industry as a deoxidizer. Second, as an electric conductor. Third, in the reduction of refractory oxides to carbon-free metals and in welding by the aluminothermic method of Dr. Hans Goldschmidt. Fourth, for culinary utensils aluminium seems to be the ideal material: it is practically indestructible and its use for this purpose should very largely increase. Dr. Richards also referred briefly to

the use of aluminium in the chemical industries, to the use of stiffened aluminium for automobile and airship construction, to the use of aluminium bronze, etc. There was quite an interesting collection of various aluminium products on exhibition.

Dr. Richards concluded that the invention of aluminium ranked with the invention of the Bessemer steel process as the most important technical achievement of the nineteenth century.

In a few graceful words congratulations were expressed to Dr. Hall by Dr. Wm. H. Walker for the American Electrochemical Society and by Dr. Chas. Baskerville for the American Chemical Society.

After the formal meeting was over there was an informal session of the Verein Deutscher Chemiker.

Nitric Acid and Nitrates from Air.

At the recent Konigsberg meeting of German Naturforscher und Aerzte, Dr. J. ZENNECK gave an interesting outline of the different processes for the fixation of atmospheric nitrogen by electric discharges through air. The paper is printed in full in *Phys. Zeit.*, Oct. 15, 1910. (See also p. 99 of this issue.)

Fig. 1 shows the arrangement of the principal apparatus for demonstration purposes. The air first enters a blower (in practice a powerful turbo-compressor is used which supplies some 20,000 cu. m of air per hour to the plant).

The air then enters the "furnace" in which a small portion

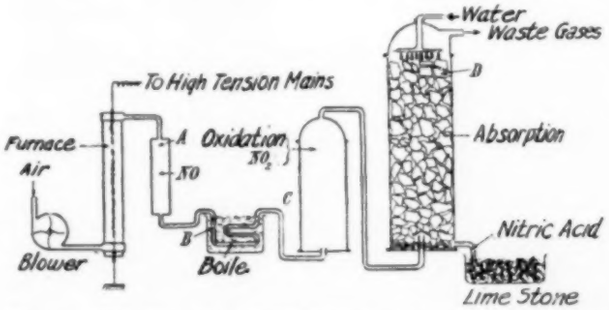

FIG. 1—APPARATUS FOR DEMONSTRATING THE MANUFACTURE OF CALCIUM NITRATE FROM ATMOSPHERIC NITROGEN.

of the nitrogen and oxygen combine to NO. The colorless gas passes off through the glass tube *A* into the boiler *B*, where the heat in the gas is utilized. In the model for demonstration purposes it is simply a water tank with copper tubes. In practice the apparatus has the same form as any large boiler. The steam produced in it is utilized for heating and evaporating.

After the gas has cooled off it has an opportunity of rest in a large vessel, the "oxidation chamber," in which the NO combines with free oxygen in the air to form NO_2.

The gas then enters a glass cylinder *D* filled with coke particles, through which water runs down, absorbing the NO_2 and forming nitric acid. In practice the nitric acid formation is carried out in very large towers.

The formed nitric acid is finally passed off into a receptacle filled with limestone for the formation of calcium nitrate.

The essential difference of the different commercial processes is to be found in details of the furnace construction. But in all cases it is an arc furnace. Convenient electrode holders for high-tension and arc experiments are shown in Fig. 2, *a* and *b*.

To get as much air as possible in contact with the arc the Pauling Brothers employ an arrangement similar to the horn lightning arrester, Fig. 3. (Details of the Pauling process, as carried out in a commercial plant at Innsbruck, Tyrol, were given in our October issue, 1909, Vol. VII, page 430.)*

Birkeland and Eyde deflect the arc, which is produced between two horizontal electrodes by magnetic means so as to form a flaming disk. The form of their furnaces is shown in Fig. 4. (See also our Vol. II, page 399; Vol. IV, pages 126, 295 and 360; Vol. VII, p. 304 and 305; Vol. VIII, p. 451.)

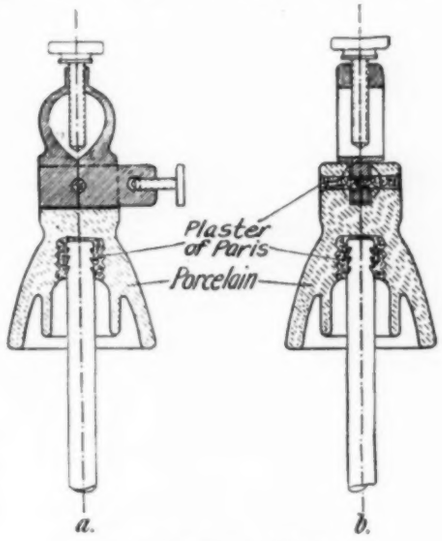

FIG. 2—ELECTRODE HOLDERS.

Petersson, Brion, and Moscicki employ a similar principle. One electrode is in form of a rod, the other is a coaxial cylinder. In the annular space between the two the arc is playing and is caused to run around continually by means of a magnetic field, the lines of flux of which have approximately the direction of the axle of the two electrodes. This is shown in Fig. 5. (See also our Vol. V, page 492.) The furnace of Moscicki

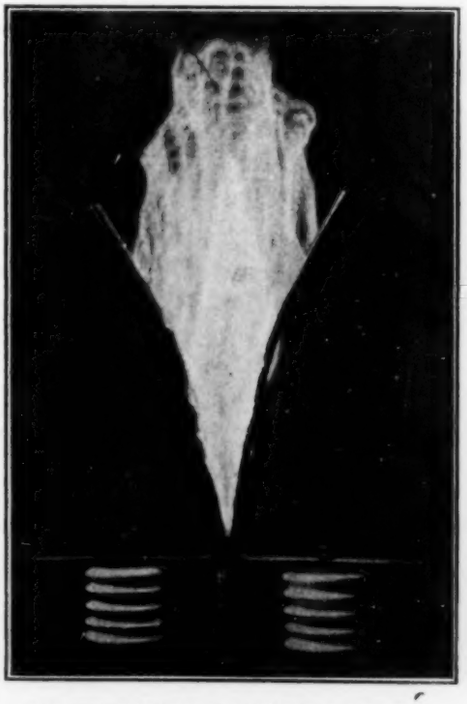

FIG. 3—PRINCIPLE OF PAULING APPARATUS.

has been further developed in conjunction with the Aluminium Industrie Gesellschaft in Neuhausen.

The furnace of the Badische Company is shown in Fig. 6. The long arc plays in the central tube and the air passes around it in a spiral path from the bottom of the tube to the top. (For further details see our Vol. VII, page 245.)

*In connection with this and the following processes the article by Lamy and Lamb on page 99 should be compared.—EDITOR.

This arc is interesting in various respects from the standpoint of the physicists. While long arcs are usually very unstable the arc is in this case maintained in a very quiet state by means of the spiral air current encircling it. Further, it is unusually long, for instance 8 m (26 ft.) at an effective tension of not over 4000 volts. Further, the conditions of pressure

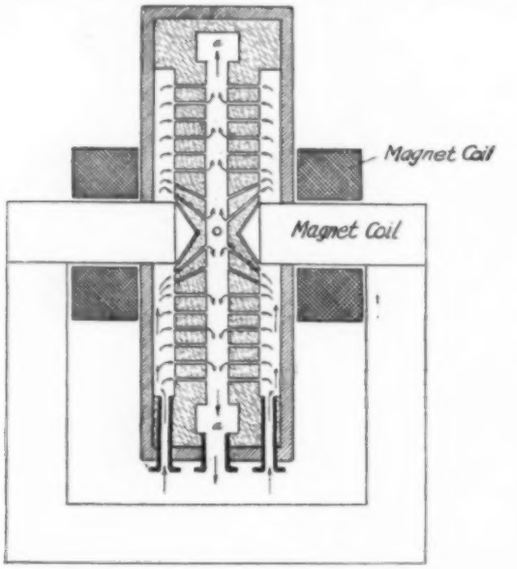

FIG. 4—BIRKELAND-EYDE FURNACE.

and temperature in such an air vortex are peculiar; in the axle of the vortex the pressure is well below atmospheric pressure. Such an air vortex acts somewhat like a centrifuge, the heavier and colder air being pressed to the outside against the walls of the tube. In consequence of this the exceedingly hot internal core with the arc is surrounded by a layer of relatively cold air.

This fact explains why Dr. Zenneck could carry out his experiment in a glass tube, although more than 10 hp were used

FIG. 5.—PICTURE OF MOSCICKI ARC.

so that considerable heat was developed. For the same reason comparatively narrow tubes can be used in the commercial furnaces with their enormous temperatures, and these tubes can be made of iron without being destroyed by the heat.

Further, the success of this type of furnace seems to contradict the hypothesis that the formation of the nitrogen oxide by the arc is a purely thermal phenomenon. According to this hypothesis it is not only necessary to heat the air to an exceedingly high temperature, but also to chill it as suddenly as possible. This latter condition is not very well fulfilled in this type of furnace.

The air to be treated is preheated by the gases leaving the furnace. The path of the air through the furnace is clearly shown in Fig. 6.

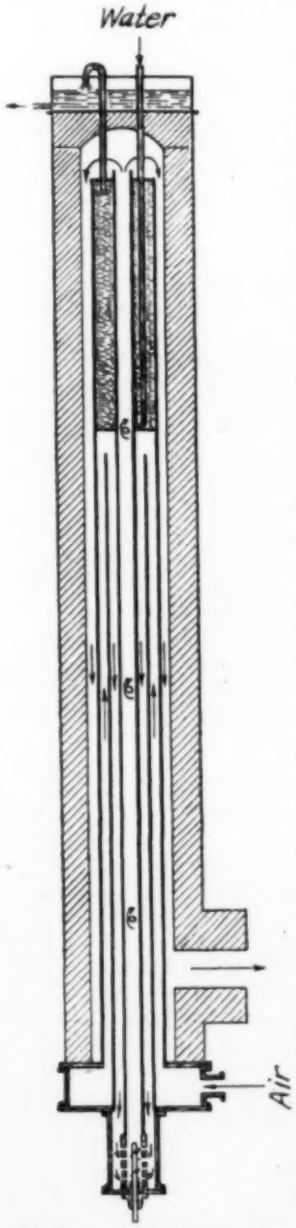

FIG. 6—FURNACE OF BADISCHE COMPANY.

The method of starting the arc is very simple. Through a little hole in the bottom a little rod of wood is introduced wound with wire so as to produce a conducting contact between the lower electrode and the wall of the tube. An arc is formed which is quickly carried upward by means of the air vortex until it is in its normal central position between the lower and upper electrodes.

Dr. Zenneck gave finally a brief outline of the electrical engineering features of the plant of the Badische Company. In the hydroelectric power plant the three phases of the large three-phase alternators are connected to the three transmission cables. A fourth thinner cable connects the neutral point of the alternator with the earth in the furnace house.

The three main transmission cables are connected each to one electrode of one furnace, the other electrode being earthed. But in each of the three main cables a large inductance coil is inserted in series with the arc. This is absolutely necessary, since without the inductance coils there would be danger of the arc being extinguished or the machine burning out. The size of these inductance coils is many times larger than ordinary inductance coils as usually employed.

These inductance coils fulfil the same purpose as a series resistor, but they represent a reactance instead of resistance—that is, they consume very little energy, since the phase difference between e.m.f. and current is very nearly 90 deg.

A novel engineering feature of the installation is the direct use of high-tension currents in the plant. The high-tension currents enter directly the works and right at the furnace electrode there is a tension of several thousand volts. This involves great danger and has necessitated special precautions.

Although the arcs are very quiet and steady once in a while an arc may be extinguished and a load of at least 1000 hp is thus suddenly taken from the generator. In the same way when the arcs are started the generators are suddenly loaded. Starting appliances, such as are used with motors, can be employed only to a small extent and are used only with very large furnaces; several inductance coils are connected in parallel and one is switched in after the other.

Further, the current and voltage conditions in the furnaces are very different from usual electrical engineering practice and great precautions must be taken in the measurements to get exact results. This becomes evident when it is found that two precision instruments, which were both calibrated by the Reichsanstalt and which agree absolutely with each other under ordinary conditions, gave readings different by 4 per cent. Commercial instruments, which are ordinarily exact by 1 per cent or 2 per cent, were found to give readings which were wrong by 12 per cent or more.

Even the connections of the instruments to the line conductor involve difficulties, as the line conductor carries a current of 200 to 300 amp and is at some thousand volts against earth.

Dr. Zenneck emphasizes these difficulties because the fundamentally important determination of the furnace efficiency depends on an exact measurement of the electrical energy. He concludes: "Since the measuring methods mostly employed in engineering fail certainly in this case, all statements in the literature on the efficiency of the different furnace systems

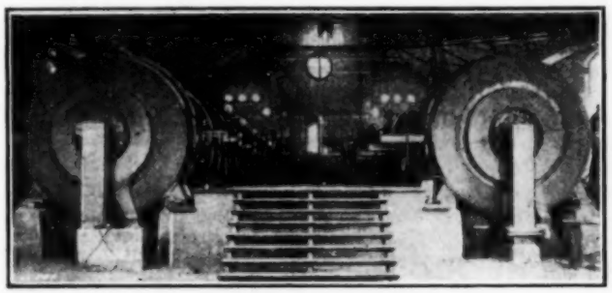

FIG. 7—FURNACE HOUSE, NOTODDEN.

should be regarded with unlimited skepticism if the statements do not prove clearly that the measurement of the electrical energy was exact."

* * *

One special difficulty of the measuring methods is discussed by Dr. J. ZENNECK in an article in *Physikalische Zeitschrift*, Oct. 15. In ordinary engineering practice measuring transformers are widely used, the measuring instruments being connected to the low-voltage terminals. But in connection with long arcs as used in the furnace of the Badische Company the employment of measuring transformers may involve unduly large errors. Dr. Zenneck claims that to be sure of the result it is necessary to insert the current coil of the electrodynamic wattmeter directly into the line conductor and to connect the voltage coil in series with non-inductive resistors across the line.

But this involves the difficulty that as the instrument is charged to a high voltage purely electric (electrostatic) forces are acting on the movable system of the instrument. Errors due to this fact must be eliminated. In the article referred to a simple method is described how to calibrate the instrument so as to eliminate this error once for all.

* * *

The works for the production of calcium nitrate from atmospheric nitrogen at Notodden, Norway, using the Birkeland and Eyde process, and especially the electric power plant, are the subject of an article in *Teknish Ugeblad*, reproduced in London *Electrical Review*, Dec. 30. The power plant contains four alternators, generating three-phase currents at 10,000 volts, 600 amp per phase, the frequency being 50 cycles per second. All four generators work in parallel on a system of busbars, which are connected with Notodden by three cables. The current is transformed by oil-cooled transformers and then supplied to the furnace, inductance coils being arranged in series.

The furnace house (Fig. 7) is a building with a floor area of about 21,500 sq. ft., constructed entirely of brick and iron. The number of furnaces is 36. "The diameter varies from 6 ft. to 8 ft., but the length is comparatively small. The electrodes are placed in the center of the furnace, diametrically opposite each other, and between them the electric arc is formed. The air is conveyed to the interior of the furnace through the walls of the chamber, which is made of firebricks, and from this chamber the nitrous gases pass through a fireproof channel. The temperature of the chamber is generally about 3500° C., while that of the escaping gases varies from 800° C. to 1000° C. The furnace can be kept going for weeks without interruption; the brickwork lasts from four to six months, while the electrodes have to be charged every three or four weeks.

"The necessary supply of air is obtained by means of centrifugal fans. The gases from the furnaces are collected and conveyed through two large pipes, each with a diameter of 6 ft., to the boiler house, where the heat contained in them is utilized for various purposes.

"From the furnace house the gases pass to a cooling house through large pipes of aluminium, placed in flowing water, and having been sufficiently cooled pass on to vertical iron cylinders, lined with acid-proof stone, where the oxidation takes place."

The information given concerning the absorption of the oxide vapors and production of calcium nitrate is meager. The production at Notodden during the first year amounted to 20,000 tons of nitrate of lime, but when all the projected works have been completed the minimum production will be 100,000 tons per year. The total capital invested when the new buildings and extensions are completed will be about $17,500,000.

Concentration of Platiniferous Copper Ore at the Rambler Mine, Wyoming.

Editorial Correspondence.

Historical.

In the winter of 1901 it was discovered that platinum occurred in the copper ore of the Rambler mine, located in Albany County, Wyoming, about 50 miles southwest of Laramie. The occurrence was first suspected from the difficulty experienced in making satisfactory assays for gold on shipments to the Denver smelters. In the usual gold assay platinum would be collected in the final cupellation, and it is difficult to see how it could have escaped detection for so long a time, for up to the date of actual discovery in 1901 nearly 4000 tons of 25 to 30 per cent copper ore had been shipped from the mine. Of this quantity, about 2500 tons had been shipped to the smelter at Encampment, Wyoming, and smaller tonnages to other plants. The incident is more striking in view of the fact the Rambler Company had never received pay for gold in its ore, as it was presumed that this metal occurred in quantities smaller than was paid for in ordinary smelting practice.

The first tests to prove the discovery were made in the laboratory of Henry E. Wood, Denver, and published by Prof. Wilbur C. Knight, of the University of Wyoming.[1] These tests showed that various copper minerals from the mine carried from 0.10 oz. to 0.70 oz. platinum per ton, while several carloads of covellite ore carried from 0.40 oz. to 1.40 oz. platinum per ton. About the same time Wells and Penfield[2] determined the mode of occurrence of the platinum to be sperrylite, $PtAs_2$. Their work was done on a sample of covellite, from an ounce of which they recovered six brilliant crystals of sperrylite.

Subsequent to this time many confirmatory tests were made by members of the company's staff, the United States Geological Survey, and others. The later investigations gave evidence of the presence, in considerable quantities, of other members of the platinum group of metals. The first commercial results were obtained from consignments of high-grade copper ore to the Chicago Copper Refining Company, Blue Island, Ill. On the assumption that all of the noble metals ultimately would be collected in the copper base in the smelting process, and later precipitated as anode slime in the electrolytic refining process,

[1] *Eng. & Min. Journ.*, Dec. 28, 1901.
[2] *Amer. Journ. Sci.*, Feb. 1902.

arrangements were made to recover this slime for further treatment. Assays of this slime showed the presence of considerable quantities of platinum and palladium, the values ranging from $5.50 to $6.50 per pound. One lot assayed 226 oz. platinum and 166 oz. palladium per ton of slime. From two portions there were actually recovered 16.74 oz. of platinum, 37.5 oz. of palladium, 5.33 oz. of gold and 38.55 oz. of silver.

Investigation of Treatment Methods.

The definite and rather extensive occurrence of the metals of the platinum group having been established, the company was confronted with the problem of ore treatment. It was apparent that much had been lost in the several thousand tons of ore shipped to copper reduction works, and that some system must be devised for saving the precious metals. A small matting furnace which had been erected at the mine had not been operated successfully, as the present appearance of the old slag dump, copper bottoms and matte shells now indicate. The failure of this plant is not surprising in view of the abundance of decomposed diorite, or so-called talc, in some of the ore, carrying 55 per cent silica and 30 per cent alumina and iron. Some of the highest grade platinum ore is especially refractory in the smelting process, one analysis showing 12.75 per cent insoluble and 52.85 per cent alumina and iron, with no lime and magnesia.

The complexity of the problem of suitable ore treatment will be better understood from a consideration of the variety of copper minerals existing in the mine, much of it low grade and requiring concentration. The Rambler has yielded, up to date, the following copper minerals:

TABLE OF MINERALS OCCURRING IN THE RAMBLER MINE.[3]

Mineral.	Composition.	Sp. Gr.
Sulphides		
Chalcocite	Cu_2S	5.7
Chalcopyrite	$(Cu_2)S_2(Fe_2)S_3$	4.2
Covellite	CuS	4.0
Carbonates		
Azurite	$(CuOH)_2Cu(CO_3)_2$	3.7
Malachite	$(CuOH)_2CO_3$	3.8
Oxides		
Tenorite	CuO	5.8
Cuprite	Cu_2O	6.0
Chalcotrychite	Cu_2O	6.0
Chloride		
Atacamite	$Cu(OH)_2Cu(OH)Cl$	4.1
Sulphate		
Chalcanthite	$CuSO_4 . 5H_2O$	2.2
Silicate		
Chrysocolla	$CuSiO_3 . 2H_2O$	2.1
Native Copper		
	Cu	8.9

These minerals occur also in diorite in all stages of alteration, even the socalled talc carrying copper and precious metals. One of the minerals, chalcanthite, is soluble in water, and the carbonates and oxides do not yield readily to concentration. Furthermore, all of these copper ores occur closely associated, one mineral sometimes coating another, and it would be almost impossible to sort them. Finally, all of them, but particularly the covellite, carry the precious metals of the platinum group very finely divided, the largest crystals of sperrylite isolated by Wells and Penfield measuring only 0.12 m in diameter. Sperrylite has a specific gravity of 10.6 and the metals of the platinum group have specific gravities ranging from 16 to 21.

The following table[4] shows the distribution of precious metals in the ores:

Identification of Sample.	Value per ton in Precious metals.	Copper.
High-grade platinum, circular drift	$36	$3.28
Hard talc, N. W. drift	12	2.75
Sulphide, black drift	26	1.45
Talc, black drift	20	5.88
Talc, covellite drift	32	12.34
Black material, black drift	88	4.65
High-grade platinum, opposite talc contact	100	75.00
White talc at contact, first level	26	5.75
Covellite, covellite drift	80	59.00
Brown talc at contact, first level	12	4.90
Precipitate above talc, covellite drift	116	68.60
General sample, stope level	26	16.40
Low grade, second level	12	2.00

From these considerations it will be seen that the Rambler ores offered a most interesting problem in metallurgy.

As a result of experiments made in 1906 under the direction of Dr. David T. Day, of the United States Geological Survey, it was found that when the ore was crushed to 2 mm and concentrated on a Wilfley table about one-half the platinum content of the ore was recovered in the concentrate, the other half being lost in the middling and tailings. The test further indicated that the precious metals occurred very finely divided and it was Mr. Day's opinion that grinding to ½ mm would result in a saving of 80 to 90 per cent of the precious metals.

In the spring of 1909 concentration experiments were carried out at the ore testing works of H. E. Wood, Denver. The results obtained on ore crushed to 20-mesh were comparable with those obtained by Mr. Day. About 56 per cent of the precious metals were obtained in a concentrate assaying 1.70 oz. platinum per ton. On reconcentrating this concentrate it was possible to obtain a precious metal product carrying 13.06 oz. platinum per ton, the ratio of concentration being 150 of crude ore assaying 0.69 oz. platinum per ton into 1 of precious metal concentrate assaying as stated above. A second test on material ground to pass a 16-mesh screen gave a concentrate assaying 1.38 oz. platinum per ton, which on reconcentration yielded a product assaying 14.62 oz. platinum per ton. This final product represented a concentration of 191 tons of crude ore into one of precious metal concentrate.

Later tests conducted by Prof. A. C. Dart, of the School of Mines, University of Wyoming; Mr. Robert A. Marr, consulting metallurgist, Union, Ore., and FitzGerald & Bennie, electrometallurgists, Niagara Falls, N. Y., indicated the possibility of profitably concentrating the low-grade copper ore by grinding fine enough to liberate the precious metals, making a high-grade precious metal concentrate and a copper concentrate carrying some precious metals. Following these opinions the company has erected a concentrating mill at the mine.

Rambler Experimental Mill.

The mill was erected adjoining the old ore bins, so that the ore could be handled from them by gravity. The mill equipment consists of crusher, rolls, trommels, classifier, Wilfley tables and canvas tables. As the plant is being operated in an experimental way, the system of treatment cannot be said to be definitely settled. In fact, the mill has been in operation just long enough to present a number of problems that have not been solved as yet, and it is expected that experimental operation will continue for some time. It will be necessary to determine what commercial products can be made most economically and in what quantity they can be produced. it will then be possible to negotiate the sale of the concentrates or consider their treatment locally, although the latter is improbable.

The mill was designed and erected by Mr. R. A. Marr, but its present operation is under the direction of Mr. Dart, who is acting as consulting engineer to the company. On page 77 is a diagram of the flow of ore through the mill.[5]

The crusher is set to crush to ½ in., and the rolls to grind to 20-mesh. The trommels are fitted with punching screens to deliver 20-mesh material. Three Wilfley tables are fed from

[3] Composition and specific gravity taken from Frazer & Brown Tables for the Determination of Minerals, 6th Edition.

[4] Assays by Fitz Gerald & Bennie. Precious metals calculated at $20 per ounce and copper at 14 cts. per pound.

[5] The abbreviation P.M. used in the diagram and following tables indicates "precious metals."

the three compartments of the classifier, and the overflow from the last compartment runs to two canvas tables. At present three products are being taken off the tables, viz., a high-grade precious metal concentrate comprising a streak of about 1 in. wide at the top of the table; a copper concentrate, and a combined middling and tailing, the last being impounded for further treatment. Owing to the lack of a regrinding machine the middling is being combined with the tailing for the present, although conditions indicate the advisability of regrinding the middling. The precious metal streak is made up of very finely divided particles, and carries the major percentage of the platinum group metals in the ore; the copper concentrate carries the bulk of the sulphides, and the middling is composed largely of 20-mesh grains of the carbonate copper ores.

The following table gives the assays of the various table

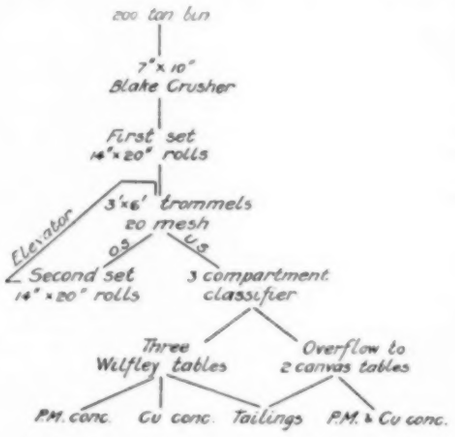

DIAGRAM OF FLOW OF ORE THROUGH MILL.

products, as made June 25, 1910, by Mr. L. S. Worthington, the company's chemist and assayer. The methods of assay will be discussed later.

	Ounces per Ton.					Per cent
Description	Total P. M.	Au.	Pt.	Ag and Pd.	Ir.	Cu.
Mill feed	1.27	....	.....	.....	.....	2.8
P. M. conc. table No. 1....	20.19	4.01	11.21	1.76	3.21	19.7
Cu conc. table No. 2.....	1.57	....	0.36	1.21	....	7.3
Middling table No. 1.....	1.15	....	.....	1.01	....	2.7
P. M. conc. table No. 2....	7.96	2.06	3.24	1.63	1.03	15.4
Cu conc. table No. 2.....	1.24	....	0.12	1.12	....	7.9
Middling table No. 2.....	1.07	....	.....	0.91	....	1.8
Mill tailing	0.90	....	.....	.....	.....	2.3

On June 30, 1910, Mr. Dart and the writer jointly sampled and assayed the various table products. The following table gives the results obtained:

	Ounces per Ton.						Per cent
Description	Total P. M.	Au.	Pt.	Ag&Pd.	Ir.	IrOs & Rh.	Cu.
P. M. Conc. table No. 1...	28.00	4.66	11.82	11.08	0.08	0.36	23.1
Cu Conc. table No. 1.....	1.86	Trace	0.06	1.26	Trace	.....	7.2
P. M. Conc. table No. 2...	17.68	2.36	4.86	10.36	0.10	Trace	22.4
Cu Conc. table No. 2.....	2.40	Trace	0.14	1.76	Trace	.....	12.6
P. M. Conc. table No. 3..	72.50	6.92	7.28	11.22	13.00	34.08	23.4
General P. M. Conc. tables Nos. 1 and 2.....	15.28	2.64	5.78	5.86	0.38	0.62	18.4

An imperfection in table No. 3 made it necessary to take the sample of precious metal concentrate periodically by hand. When the sample was not being taken the precious metals ultimately combined with the copper concentrates and were, therefore, not included in sample six of the above table. For the same reason no assay appears for the copper concentrate of table No. 3. In preparing the samples for assay no metallics were observed on a 100-mesh screen. Of the precious metal concentrates 33 per cent of the sample from table No. 1 passed a 100-mesh screen; 25 per cent of sample No. 2 and 90 per cent of sample No. 3 were also 100-mesh and finer. It will be observed that the richness of the sample varies directly with the fineness, the finest material being the richest.

Assay Methods.

In view of our comparatively meager knowledge of assay methods for metals of the platinum group, it would be unfair to all concerned to present the foregoing figures without an accompanying statement of the methods employed. The assay for platinum is usually made on sands containing metallics, or on alloys of the metal, and it is seldom that ores carrying the platinum group metals are presented for assay. The figures given in the foregoing tables do not represent *direct weights* of the metals indicated, but are calculated from *differences in the weights* of the buttons after successive partings with different acid solvents which are supposed to extract certain metals from the button.

The procedure is based on the following statements made by Lodge[6] and Fulton.[7] In an alloy of the precious metals,

(1) Sulphuric acid will dissolve silver and palladium together;

(2) Nitric acid will dissolve platinum when the ratio of silver to platinum is 12 or 15 to 1, or when the ratio of silver to gold to platinum is 15:2:1;

(3) Dilute aqua regia, 1:5, will dissolve gold;

(4) Concentrated aqua regia will dissolve iridium, and,

(5) The final residue insoluble in the foregoing treatment will be iridosmium and rhodium.

The method of assay used at the Rambler mine at present is as follows: Crucible fusions of ½ A.T. of material are made in duplicate. To one crucible is added silver in quantity about four times the weight of the precious metals believed to be present. As there is considerable copper and precious metal in the lead button, the cupellation is carried out at a high temperature, and if the resulting bead is large it is recupelled with an addition of lead to clean it thoroughly. The high temperature employed probably volatilizes any osmium and rubidium as oxides, so that these do not remain in the bead. The beads are dark gray in color, flat and rough.

The button to which no silver was added is weighed and calculated as total precious metals. The button alloyed with silver is parted with sulphuric acid and weighed. The difference in weight between this residue and the total precious metal button is calculated as silver and palladium. No further use is now made of this residue, unless it should be needed to replace a button lost in subsequent operations, or to check later results. The total precious metal button is now alloyed with silver, using 12 to 15 times the weight of residue from which silver and palladium have been dissolved.

This button is parted with nitric acid, specific gravity 1.20, and weighed, recording the differences in weight between it and the next preceding weight as platinum. The residue is then warmed with dilute aqua regia, 1:5, which will dissolve the gold, the weight of which is calculated by difference after another weighing. Finally the residue is boiled with strong aqua regia, dissolving iridium, which is estimated from the difference in weight between the residue and the next preceding it. The final weight gives iridosmium and rhodium. In alloying the cupelled buttons with silver in the first stage of the work it is noticed that small dark spots appear on the molten button, finally settling on the bottom when cold. These are probably iridium, which is exceedingly difficult to alloy. In making the partings it may be necessary to repeat the operation if for any reason it is feared that solution is incomplete, and this may be continued until a constant weight is obtained.

There are points in this procedure about which questions can be raised. It is doubtful whether the solvents used are without action other than that attributed to them, so that the final assay figures for any one of the metals may be high or low, due to incomplete solution of one of the constituents of the residue, or to partial solution of a constituent assumed to be insoluble. In any event, the total precious metal button can usually be relied upon, and the various constituents are reasonably correct.

[6] Notes on Assaying.
[7] A Manual of Fire Assaying.

This does not obviate the necessity or desirability of more exact procedure, and it is hoped that the production of high-grade precious metal concentrate at the Rambler will stimulate research in this direction.

Mention has been made of smelting operations carried on formerly at the Rambler mine. A matte smelter was erected in 1902 and operated for six or eight months, but finally abandoned as unsuccessful. Probably suitable fluxes were not available, and as the ore does not carry much iron, it was difficult to smelt. Samples of matte and copper bottoms taken from material on the dump showed, respectively, the presence of 17.00 oz. and 32.52 oz. of precious metal per ton of such material, thus showing the probability that precious metals have always occurred in the ore and have been lost in most of the previous shipments to copper reduction works. The data further indicate the possibility of gathering the precious metals in copper matte and blister copper. These facts, together with some work done for the Rambler company by the Chicago Copper Refining Company and FitzGerald & Bennie lead to a suggested system of metallurgy. The work referred to consisted in reducing some of the Rambler high-grade ore, producing blister copper. This was analyzed by FitzGerald & Bennie, with the following results:

Copper	91.1 per cent.
Precious metals	3.91 oz. per ton.
Silver	6.42 oz. per ton.
Lead	2.17 per cent.
Nickel	0.60 per cent.
Antimony	1.95 per cent.
Iron	0.90 per cent.

The lead was probably gathered from previous operation of the furnace, as it does not occur in the Rambler ore in any appreciable degree. Nickel has been identified in Rambler ores and antimony also occurs in small quantities, some authorities giving it as a constituent of sperrylite. It was from the electrolytic refining of such blister copper as this that the anode slime previously referred to was recovered and subsequently refined by Baker & Company and the S. S. White Company.

In consideration of the foregoing data it would seem as though a rational system of metallurgy for Rambler ores would be the following: Direct smelting of the high-grade (25 to 30 per cent) copper ore, producing a matte subsequently to be blown to blister copper. Concentration of the low-grade (2 per cent) copper ore, recovering a high-grade precious metal concentrate which could be refined chemically, and a copper concentrate which could be smelted, blowing the resulting matte to blister copper; or, all of the valuable concentrate could be combined for smelting. The copper obtained from all smelting operations could be refined electrolytically and the anode slime containing the precious metals could be treated chemically. The copper carbonate and oxide minerals in the low-grade milling ore, which pass into the tailing pond, could be leached with the mine water, which is quite acid and also carries some copper in solution. The copper dissolved in this process could be precipitated on scrap iron, but the precious metal in the tailing would, for the present, have to await the development of some method of recovery. Or it might be found that it would pay to smelt 2 per cent copper tailings if the silica were not too high and the precious metal content were sufficiently valuable. The question of transportation would play a large part here. It will be apparent from the assays previously given that the concentration of the copper values in the low-grade ore would scarcely be profitable were it not for the precious metals recovered, but since the copper lost in the tailing and middling is almost all in a soluble form, the process outlined is expected to prove profitable and may be adopted.

The writer is indebted to Mr. W. J. Sherwood, mine manager for the Rambler Copper & Platinum Company, for permission to publish the data given herewith, and to Mr. A. C. Dart, consulting engineer, for courtesies extended.

Exploitation of Peat Bogs.

The exploitation of Canadian peat bogs for the production of fuel for domestic and industrial purposes was the subject of an interesting address, delivered last year before the American Peat Society by Dr. Eugene Haanel, Director of Mines, Ottawa, Ontario, Canada. This address has recently been published in form of a pamphlet, from which the following extracts are taken.

A former extended investigation of the manufacture of peat fuel, carried out by the Department of Mines of Canada, has demonstrated, firstly, that for the economic production of fuel from peat machinery driven by power must be substituted as far as possible for manual labor; and, secondly, that processes so far invented for removing the water content of the peat by pressure and artificial heat have not led to commercial results, and after trial have been abandoned.

Two promising processes employing artificial heat in the production of peat fuel have given disappointing results.

The Electropeat Syndicate erected an extensive plant at Kilberg, in the County of Kildare, Ireland, for the production of a substitute for coal from peat, which they expected to sell at six shillings per ton. No money was spared in trying to make the experiments a success, and most expensive machinery was put down. Experts were brought from Germany to look after the work under the superintendence of Captain Verey, R. E. Large shipping companies on the strength of samples submitted, and representations made to them, promised some very large contracts for fuel, but the company, at the end of over one year's experimentation, found that its process was a failure. The peat made by it looked like coal and burned well, but was hygroscopic, and after a short time crumbled to powder. All efforts to overcome this defect failed, and the project was permanently abandoned.

The Elkeberg process of the wet carbonization of peat is exceedingly ingenious, and has aroused great expectations, but is still in the experimental stage after the expenditure of 200,000 kronen in Sweden to place it upon a commercial basis.

The endeavor to accomplish *economically* by artificial means in a *short time* what has been accomplished by nature in exceedingly long periods of time, the change of peat into a substance similar to coal, has so far apparently not been attended with success. Dr. Haanel would not like to say that it cannot be done, since it is unsafe to make any statements regarding the possibilities of future achievement, but at present the outlook in this direction is not encouraging.

The proper course for Canada is, as Dr. Haanel points out, to introduce such processes and such machinery as have proved successful and are now in actual commercial operation in Europe.

To re-establish the confidence of the people of Canada in the value of peat as a domestic and industrial fuel and to stimulate renewed activity in the development of the great peat resources of Canada the government has acquired 300 acres of peat bog, with an average depth of 9 ft., for the purpose of manufacturing peat fuel on a commercial scale, and by a method which has proved successful in European practice. At this plant interested parties have an opportunity of ascertaining for themselves the working of the bog as well as the suitability of the peat fuel produced. The capacity of this plant is a production of 30 tons per day. For a large commercial plant mechanical excavators should replace the manual labor employed at this plant, if the bog to be exploited is suitable for this class of labor-saving machinery.

The plan at Alfred is to serve as a model of a successful process, and not for the production of peat fuel on an extensive scale. It was expected, however, to manufacture during the first season about 2000 tons of peat fuel, part of this to be used in their peat-gas producer at Ottawa.

There is nothing artistic about the appearance of the fuel produced at this bog. It has not the regular geometric form

of briquettes nor their smooth exterior, but it serves the purpose for which fuel is intended as well as briquettes, and has the advantage of being low in cost of manufacture.

Allowing 140 days for a season's operation, the cost per ton of air-dried machine peat, including interest on capital invested, amortization, oil and repairs, is given by Dr. Haanel as follows:

Cost of fuel on the field	$1.40
Cost of fuel stored in shed	1.65
Cost of fuel loaded on car	1.65
Cost of fuel in stack	1.70

By the employment of mechanical excavators and the manufacture of peat on a large scale, the cost of production per ton should be considerably less than the figures here given.

The objection to the air-drying process practised at this plant is that it is not a continuous process, that it can be worked only during the summer months, and that the amount of fuel which can be produced during one season is dependent upon weather conditions. But air-drying is employed successfully on a large scale in Europe, and especially in Russia, and the weather conditions in Canada are at least as favorable, if not more so.

The economy which may be effected by the use of peat in Canada is readily understood when it is stated that in Ontario and Quebec the average price of anthracite with a high percentage of ash is $7.50, and in Manitoba, $10.00 per ton, whereas air-dried machine peat, containing only about 5 per cent of ash can be manufactured at a cost of considerably under $3.00 for an amount having the same calorific value as a ton of the anthracite which must be imported. This could be sold at places conveniently situated as regards transportation facilities, and not too far from the place of manufacture for a little more than one-half the price paid for anthracite in Ontario and Quebec, and for a little more than one-third of the price paid for anthracite in Winnipeg.

Air-dried peat is not alone an excellent fuel for domestic use, but for the production of power it proves an ideal fuel in the peat-gas producer, which is to-day as reliable and efficient in its operation as the coal-gas producer. Dr. Haanel did not hesitate to say that it is an ideal fuel because the peat from most bogs is free from a clinkering ash and yields on combustion a fine, white residue, which readily allows of the thorough cleaning of the fire, and the property of not fusing or caking in the producer assures regular operation.

Moreover, since gas leaves the producer with a high degree of sensible heat, which must be cooled to the temperature of the atmosphere before being used in the gas engine, it is exceedingly important that as much of this sensible heat as possible be utilized in the producer itself, in order to increase its thermal efficiency. This is accomplished in coal-gas producers by the introduction of water vapor which passes through the incandescent fuel with the air supplied for combustion. This water vapor is decomposed, yielding hydrogen and oxygen. The latter combines with the carbon of the fuel, forming carbon monoxide.

This chemical reaction absorbs a large amount of heat and lowers the sensible heat of the gas, but the heat absorbed in liberating the hydrogen is to a large extent restored and utilized when the gas enriched by hydrogen is burned in the gas engine or other apparatus. With peat containing from 25 to 30 and more per cent of moisture the latter is sufficient to accomplish all that is required without the introduction of water in the producer from an outside source.

To demonstrate the value of peat for the production of power for industrial purposes a modern German peat-gas power plant has been erected by the Department in Ottawa. Its capacity is 60 hp, and consists of a double firezone Koerting peat-gas producer, with the necessary gas-cleaning apparatus, and a Koerting four-cycle single-acting gas engine direct connected to a Westinghouse 50-kw, direct-current generator.

The consumption of peat per brake horse-power is expected to be in the neighborhood of 2 lb., definite figures not yet being available.

The erection of gas producers designed for the recovery of by-products is not recommended except in localities where such by-products would command a ready and profitable market. In Canada it is far more economical to aim at the complete gasification of all the heat elements in the fuel.

Nucleus Action and Grain Growth

By HENRY M. HOWE.

The purpose of this note is to consider the explanation which Mr. E. Cohen gives of the interesting cases of nucleus action which he has observed.[1] He finds that tin which has been dulled in a certain way, when pressed against bright cold worked tin, such as tinfoil, or tin plate, inoculates these latter with like dullness, and that these infected spots thus set up can carry the infection to still others, slowly at room temperature (§4) and more rapidly at higher temperatures. (§§ 6, 13, 14.) What is the exact nature of this infection? The bright surface which he dulled in most of his experiments had been made by cold rolling or like action, and he explains the infection as an undoing of this cold rolling, reasonably calling the cold-rolled state metastable, and the non-cold-rolled state, that is, the dulled state to which infection brings it, the stable state ("forciert" and "nicht forciert," which we may translate as "overstrained" and "not overstrained," "ecroui" and "non-ecroui").

A prominent effect of overstrain, by whatever form of cold-working it is caused, is to distort the individual crystalline particles of which the metal is normally composed. Most metals, iron included, belong to the cubic (monometric) system; and the distortion incident to overstrain, of course, changes by different amounts the length of the several axes of any given crystalline grain, so that these grains, normally "equiaxed," become "inequiaxed." So we often refer to the "inequiaxing" caused by overstrain, and the "equiaxing" caused by annealing, one feature of which is that it restores the crystalline grains or particles to their normal state. But as tin belongs to the tetragonal (dimetric) system, with axes normally unequal, "equiaxing" and "inequiaxing" are not strictly applicable. Let us rather speak of "crystalline distortion" or "abnormal axing" and "crystalline readjustment," or "normal axing."

Applying these ideas to Mr. Cohen's experiments the initial bright state of his tinfoil and tin plate is one of crystalline distortion or abnormal axing, the diseased state; and the dulled state is one of cure, readjustment, or normal axing, in spite of the fact that here the cure is, as far as the practical applications of the tin are concerned, far worse than the disease.

The question which I wish to consider is, whether under Mr. Cohen's various conditions we can properly regard crystalline distortion and readjustment as the chief causes of the phenomena. Let us consider these phenomena.

A first case is that of the dulling of tin plate by reheating. Simple exposure to 161°, 176° and 184° did not cause dulling, though like exposure when coupled with contact with dulled tin did, and though Behrens found that bright faces of very greatly overstrained tin dulled at 110° and very rapidly at 150° (§17). This case falls readily into line. The molecular mobility caused by heating, and the nucleus action of dulled (readjusted or normal axed) crystals, are clearly allies, jointly favoring the readjustment (dulling) of distorted crystals. Hence it is but natural that their joint action should cause the crystalline readjustment of which dulling is but an evidence, even when heating alone fails to; and that the greater instability of Behrens's greatly distorted grains might cause them to submit to the tendency to readjustment, which heating to 150° or even only to 110° sets up, even though the less unstable grains of Cohen might defy this tendency to readjustment even at

[1] *Zeitschr. phys. Chemie*, 1909, **68**, *p.* 214. See also this journal, Vol. VIII, p. 510.

184°, unless it were reinforced by alliance with the nucleus action of dulled and normal-axed particles.

A second case is that careful oblique hammering on the bright surface of tin plate so as to destroy the polish sets up the dulling, which in turn dulls tinfoil pressed against it at 100°, setting up centers whence the dulling spreads. (§ 21.) This case, too, falls into line, if we assume that an exaggeration of the overstrain creates a degree of instability, an abnormality of axing, so great that the readjustment and normal axing occur, as in Behrens's case, without nucleus action. In short, the instability, when exaggerated, from being metastable becomes labile.

A third case may be considered. The watered (moiré) surface of etched tin plate dulls a bright tin plate when pressed against it at 184°, though two bright tin plates thus pressed do not dull each other, and though the watered surface is not affected by contact with dulled tin. (§§ 20-21.) To bring this into line we have to suppose that this watered look caused by etching is due to the readjustment and normal axing of the crystals, so that etched tin and dulled tin are alike in being normally axed.

A fourth case is less clear. The bright surface of a quietly frozen tin ingot is readily dulled by contact with dulled (infected) tin, whereas the surface and even the interior of a tin ingot shaken during freezing has the dulled look of infected tin, and has the power of dulling bright tin. In order to bring this into line we seem forced to assume that the bright surface of the quietly frozen ingot is in effect overstrained. When the metal is first poured freezing is likely to take place first at the bottom and sides of the ingot in contact with the cold mold. Later a thin skin freezes over the upper surface; and though this skin remains bright, its crystalline structure may misfit that which, spreading up quickly from bottom and sides, approaches it from below, so that the very skin may find itself in unstable crystalline equilibrium, not because itself abnormally axed, but because discordantly axed with regard to the mass underlying it. Or its own surface tension may so interfere with the crystallization of the very skin as to make it unstable. Because of this instability the surface may recrystallize and dull when to the influence of the underlying discordant crystals, or of its own instability, is superadded the influence of the dulled tin pressed on it from above.

But this explanation may be accused of bearing too much good will; and we find ourselves left with a desire either to clinch it by other and corroborative evidence, or to seek some further explanation. The amorphous state may give us the key.

In this connection there is great interest in Mr. Charpy's[1] discovery that previous overstrain accelerates enormously the growth of crystalline grain (Stead's brittleness) which occurs when very low carbon steel is held between 650° and 800°. Thus two like pieces of very low-carbon and moderately phosphoric steel, one wire drawn and the other not, were held in this range of rapid growth between 650° and 800°. After this the grains of the wire-drawn piece were about 10 times as wide as those of the other, implying a difference in volume of 1000 to 1. Steel which when wire drawn could be bent considerably without cracking, after later heating at about 650° became so brittle that it broke when dropped on the ground. Thus, though one effect of reheating wire-drawn steel is truly to anneal it, by normal axing and relieving stress, and perhaps in other ways, yet in case of very low-carbon steel an undue exposure to this very annealing temperature may cause a grain growth far more harmful than the abnormal axing and the stress which it removes. The cure of two diseases may here give rise to a third worse than those two.

That this growth should be rapid in this very low-carbon metal so heated is indeed natural. In the presence of much carbon the growth of the ferrite, whether alpha below Ac2 or beta between Ac2 and Ac3, is naturally impeded by the presence of the cementite. In very low-carbon steel the relative freedom from cementite removes this obstacle to growth. Of course, if the temperature rose past Ac2 so that the iron became beta the particles of this modification would be nascent during the change, and this nascency might well lead to extremely rapid crystalline growth. That the coarse grain should be effaced when, on rising past 900°, the iron changes into the gamma state is natural enough. The new state might well form new crystals of its own.

The reason for the acceleration of the crystalline growth caused by overstrain remains to be determined. Of course, the abnormally axed overstrained crystalline particles are initially in unstable equilibrium, so that to the impulse toward crystalline readjustment due to the mobility of the high temperature, say 750°, is added the impulse due to the initial instability, resulting in a greater rapidity of readjustment. One could hardly foresee that this should lead to a more rapid growth of grain; but the fact that it does is not unnatural, and may perhaps be likened to the lessening of friction which motion causes. Just as the mechanical friction of motion is far less than the mechanical friction of rest, so the anti-crystallogenic friction may be less in the overstrained metal because of the added rapidity of molecular motion caused by the instability due to initial overstrain.

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[1] *Rev. Metallurgie mem.*, 1910 **7**, p. 655.

The Death of Dr. F. A. Kjellin.

We regret to report the sudden death of Dr. F. A. Kjellin at the age of 38 years on Dec. 30, 1910. His name will be forever remembered for his connection with the development of the electric induction furnace. While others before him, especially Mr. E. A. Colby in this country, had invented and tried an induction furnace, Kjellin was the first to use it on a commercial scale, and he did important pioneer work in electric steel melting and refining.

THE LATE DR. F. A. KJELLIN.

Frederick Adolf Kjellin was born in the county of Vardinge, not far from Stockholm, Sweden, on April 27, 1872. He was educated at the Royal Technical Institute in Stockholm from which he graduated in 1893. During subsequent years he carried out a great deal of experimental work in chemistry and metallurgy, and worked out a new method for the electrolytic precipitation of zinc, on which method he obtained a patent. During the years 1893-1895 he assisted Professor Wiborgh in developing the well-known Wiborgh termofon for the determination of high temperatures. From 1895 to 1899 he was engaged in developing the Lux light now used extensively.

In the year 1899 he began at Gysinge his work on the use of the electric induction furnace for steel melting. In 1904 he became connected with the Swedish Gröndal-Kjellin Company, in the capacity as chief of the electrometallurgical department. During the last few years he acted as consulting engineer in this department. The honorary degree of Doctor of Philosophy was bestowed upon him by the University of Upsala in 1907.

The Manufacture of Ammonia-Alkali in England.

BY WM. MASON.

There is no branch of chemical industry in which the principles of engineering science have been so profitably applied as that of the manufacture of ammonia-alkali. The world's production of this article, which in 1883 was only 160,000 tons, has steadily increased until at the present time it is estimated to amount to upward of 2,000,000 tons per year, about nine-tenths of which is said to be produced by the Solvay system. The English Solvay works of Messrs. Brunner, Mond & Company, Ltd., in the Cheshire salt district, are the largest in the world and since the year 1881 have continued to show the most magnificent financial results.

It has been thought by some that the Solvay system was a secret method, but that is not so. This system works the method first made known by Schloesing and Roland in 1854, which is a modification of that practised by earlier workers, such as Dyar and Hemming, who obtained the first patent for this process in 1838. It is represented by the following equation:

$$Na\,Cl + NH_3 + CO_2 + H_2O = Na\,HCO_3 + NH_4\,Cl.$$

The ammonia-alkali process is so simple, so unlike the Leblanc process, that any one with only a little experience in chemical manipulation can produce sodium carbonate by it.

Solvay's first patent in 1863 was for plant, and a few years afterward he established works at Couillet, in Belgium, making in the first year 179 tons of soda ash. He was the first to make the manufacture of ammonia-alkali a financial success, increasing the production each year, until in 1874 he manufactured upward of 4678 tons, when it was introduced into England by the late Dr. Ludwig Mond with a success that is without a parallel in the history of chemical industry. The English Solvay works have paid an average dividend for the last 28 years of 31 per cent, the lowest in 1881 of 19 per cent, and the highest of 100 per cent in 1893.

Notwithstanding such success it can be shown that in the works of this successful company, when they were annually making a profit of nearly £200,000, in trying to improve the chemical side of this manufacture, they adopted and persistently worked a method which must have added at least 10 shillings per ton to the cost of production.

It has often been stated that in the carbonating part of this process a quantity of sodium bicarbonate remaining dissolved in the residual chloride liquors would always be lost, and to recover this bicarbonate Solvay suggests in one of his patents the placing of salt in the carbonating tower; but ordinary salt, as supplied to chemical works, contains impurities equal to 0.8 per cent, which increases the percentage of insoluble matter in the finished soda ash. In 1878 Dr. Mond obtained a patent for a method for purifying the brine by adding to it a kind of water-softening mixture; by this method he obtained a very pure salt. This process was worked for some time and then discontinued, ordinary salt being afterward used.

Now there is no evidence whatever to show that any of the salt added in this way was ever decomposed, or was necessary to prevent the loss of bicarbonate in these liquors. For this purpose about 10 cwts. of salt per ton of soda ash produced was required; this salt contained 10 per cent of water, which diluted the liquors, the dilution being further increased by 50 per cent more water being required to wash out the excess of salt from the bicarbonate. This water carried away bicarbonate in solution, thus making more liquor to be passed through the ammonia still, which increased very considerably the cost of manufacture.

Below are given some analyses of the residual liquors: No. 1 from Solvay towers having had salt placed in them; No. 2, from another works which did not use salt in the carbonating apparatus, while No. 3 gives the analysis of liquors from Solvay towers after finding out that salt was not required to obtain a residual liquor from the carbonating tower, with little or no sodium bicarbonate in solution:

No. 1. SOLVAY TOWER LIQUOR WITH SALT ADDED.

Percentage alkalinity calculated as NH_3	1.49	1.19	1.61	2.22
Ammonium chloride	16.53	17.12	16.42	14.29
Sodium chloride	13.11	12.98	12.29	14.98

No. 2. CARBONATOR LIQUOR WITHOUT SALT.

Percentage alkalinity calculated as NH_3	1.65	1.87	1.56	1.63
Ammonium chloride	18.88	17.97	18.08	17.33
Sodium chloride	6.88	7.28	7.74	8.00

No. 3. SOLVAY TOWER LIQUOR WITHOUT SALT.

Percentage alkalinity calculated as NH_3	2.02
Ammonium chloride	17.22
Sodium chloride	8.45

Since for every equivalent of ammonium chloride produced an equivalent of sodium carbonate is precipitated as sodium bicarbonate, analyses Nos. 2 and 3 clearly show that slightly better results were obtained than by the addition of salt.

Although this process is an English one, it was not worked in England until the year 1874, when works were built at Northwich, and a year later at Sandbach. In 1874 Solvay manufactured upward of 4678 tons and it is evident that the works mentioned above were of the same capacity. They are very clearly described by him in the specifications of patents No. 999 and No. 1904, of 1876. The plant at Sandbach consists of lime kilns, four ammonia stills with heater and absorbers, five carbonating towers, four drying machines, besides other vessels. When working this plant showed a profitable return, but it was almost impossible to keep it continually working because of its extremely complicated parts, chiefly in the ammonia stills, the connecting pipes becoming blocked and the changing valve leaking. Moreover, the lime, being charged unslaked, was very badly burned.

Through these causes it might safely be said that the plant did not work more than two-thirds of its time, yet with all its drawbacks this plant was working in the year 1881. This plant could never have competed favorably with the Leblanc process and the ammonia alkali industry was in a most critical condition when the late Dr. Ludwig Mond, from the capacity of the ammonia distilling apparatus of this plant, designed the vertical still afterward patented by him in the year 1883.

This still is a fine example of chemical engineering, as it shows how the principles of engineering may be applied in the construction of chemical plant; it was such an improvement over the four-still arrangement that the daily output was increased by more than one-third, that is, from 13 tons to more than 21 tons, without increasing the other parts of the plant; in fact, this vertical still made the plant almost continuous, having only to be stopped a day in about six weeks.

This plant was duplicated and worked most successfully for many years. A vertical still had been used in distilling ammoniacal liquors long before this time, and it seems strange that it was not used at first, as it was much cheaper to construct. Solvay describes one in his specifications of patents for 1876.

The carbonating tower patented by Solvay in 1872 is also an apparatus which shows a considerable amount of engineering skill and may be considered his best contribution to this particular process and it still remains a part of the present-day plant with the addition only of an internal cooling coil.

Although the patent protecting this Solvay carbonating tower, which was so ingeniously designed and which produces the most perfect results in working, became void in 1886, another works was established at Runcorn which contained a plant much simpler in construction than that of Solvay; in fact, it consisted chiefly of large cylindrical vessels made of boiler plate to the number of about 30. It was worked as follows:

The ammoniacal gases from the stills, without cooling, passed into vessels called saturators which contained salt-brine and into which solid salt was also charged by means of a cage run-

ning down the length of the saturator, so that brine was made and saturated with ammonia in one operation. The ammoniated brine so produced was similar to that obtained in the Solvay system.

The great drawback of this method was that much salt became deposited in the saturators, sometimes to the depth of 12 ft. or more, which made it difficult to estimate the ammonia in them, a matter of the greatest importance in this process. The stills in this department on several occasions recovered upward of 620 lb. NH_3 in the hour.

From the saturators the ammoniated brine, after cooling and settling, was passed into vessels called absorbers, two in number, so that while one was being filled with ammoniated brine the other was working as a part of the carbonating plant. The carbonating plant consisted of three vessels, each containing about 600 cu. ft. of ammoniated brine. The carbonic acid gas from the lime kiln, after being washed and cooled and at an initial pressure of 38 lb. per square inch, passed through these concentrators, through an absorber, then through a tank containing brine and lastly through a vessel containing acid.

Five carbonators were worked off during the day of 24 hours and produced about 10 tons of soda ash. The washing of the soda bicarbonate was so carried out that about 20 per cent of it was washed away and lost. Analyses of residual liquors from above carbonators have already been given above (Table No. 2).

By the Solvay towers the 600 cu. ft. of ammoniated brine could have been worked off in less than three hours, that is, in the state of carbonation as it passed into these carbonators, and at least 15 tons of soda ash might have been produced instead of 10 tons. The plant was supposed to contain a quantity of ammonia equal to 55,000 lb. NH_3, a quantity quite sufficient for a daily output of 35 tons of finished soda ash. Sixteen tons of limestone were also burnt per day. A particular feature in this plant was that the vessels were so placed that the liquor passed by gravity from vessel to vessel so that pumps were not required. This works was not a success.

At Middlewich, in 1889, a plant very similar to the one already described was erected, but on a much more extensive scale. It contained two large ammonia stills very similar in construction to the Mond still mentioned in the Solvay system, except that the superheater was connected by pipes instead of being built in one column. It also contained four sets of carbonators similarly worked as at Runcorn, and four lime kilns. With a little ingenuity this plant might have produced at least 600 tons of soda ash per week; with the whole plant working only 390 tons were made and with half the plant but 200 tons of soda ash were produced per week.

To illustrate this point two examples will be given. When the whole plant was working and contained a quantity of ammonia equal to 123,000 lb. NH_3, 104 tons of limestone were burnt, 16,000 cu. ft. of residual liquor distilled, 15,000 cu. ft. of brine used; 55.8 tons of soda ash were produced in the day of 24 hours. To effect this about 20 carbonators were worked, each containing 840 cu. ft. of ammoniated brine.

Another example when only half the plant was working and contained a quantity of ammonia equal to 57,000 lb. NH_3; 6600 cu. ft. of residual liquor were distilled, 6600 cu. ft. of brine used; 23.68 tons of soda ash were produced; about 10 carbonators were worked off. The loss of ammonia in the working of this plant varied from 3 per cent to 15 per cent, that is, parts of ammonium sulphate per 100 parts of soda ash. This works in the year 1893 with a capital of some £180,000 lost £10,689 on the year's working.

Another works with about the same amount of capital paid only 7½ per cent dividend. In fact, three works of very similar construction were bought up by Messrs. Brunner, Mond & Company, Ltd. If these works had used the Solvay tower or had used more engineering skill in the construction and working of their plants much more favorable returns would have been made.

Similar intermittent plants have also been worked in different parts of Germany about 30 years ago, all of which are said to have failed. None of these intermittent systems could ever succeed against the Solvay system and the price of alkali need not have been lowered to £2.175 6d. per ton, as it must have cost more than £4 per ton to produce soda ash by them.

Islip, Oxford.

Cyanidation of Ores.*

By Dr. Wilbur A. Hendryx.

The chemistry of the cyanide process is so well known and universally accepted as commercially correct that I will pass it over, as the object of this paper is to deal with the mechanical handling of ores and the different apparatus employed.

For convenience sake I will divide the ores into four classes:

1. The silicious ores, requiring amalgamation and cyanidation only.

2. Ores containing, besides silica, iron, lead, zinc and copper sulphides, etc., in which amalgamation and cyanidation recover a large percentage of the gold and silver contents. (The precious metals being physically combined.)

3. Ores containing sulphides in which a portion of the gold and silver is chemically combined and at the same time carry a larger percentage of cyanicides, which can be removed by concentration before subjecting them to cyanidation.

4. The sulpho-tellurides, which up to the present time require roasting to put them in a condition to be transferred from the fourth class to the first.

It has been my practice for years to grind all ores and tailings which are to be cyanided to a sufficient degree of fineness to permit the solutions to come readily in contact with the metal which is to be extracted, whether it was ¼ in. or 200 mesh.

I will first refer to the mechanical devices and arrangement of the same for the class of ores arranged under the head of class 1.

I preferably use two rock breakers and deliver to the stamps a product not larger than 1 in. The stamps most commonly used weigh 1050 lb. each and drop 100 to 105 times a minute. A 12-mesh to 16-mesh screen is used, thereby giving from five to eight tons to the stamps per 24 hours, with approximately four tons of solution to one ton of ore. I use the word "solution" instead of "water" because I invariably find where wet crushing is resorted to a weak alkaline solution of cyanide (1/40 of 1 per cent or less) increases the amalgamation of the gold and silver, except where it is prohibitory owing to the presence of a large amount of cyanicides, necessitating concentration, as in the case of ores I have placed in the third class. Not only is amalgamation increased by crushing in cyanide, but considerable gold is put into solution at the battery; consequently, as a general rule, the best practice is to begin cyaniding at the battery.

Dropping the subject of wet crushing for a moment and taking up the ores of class 4, namely, sulpho-tellurides which require roasting, my experience has led me to believe that the most economical apparatus on the market to-day for reducing the ore to the proper mesh dry, after the rock breaker, before it goes to the roasting furnace, is that type of mill known as the Bonnot mill or Kominuter. While they are not adapted to wet grinding they are particularly efficient in dry grinding. Of course, there are many varieties of rolls, some of which are also adapted to dry grinding.

Referring again to wet crushing with stamps I believe up to this point we have reached the greatest capacity and economy. The next step after the stamps is amalgamation if coarse gold is present. After the plates a dewatering sizer is used so that the ore which has been ground fine enough to permit the solutions to come in contact with the gold and silver is classified out and taken to a storage settler, preferably a Hendryx combination agitator and filter, which in that instance acts as a

*Abstract of a paper presented before the Scientific Society of Colorado.

storage settler (up to the time of its having received its proper charge of solids, when it is then used as an agitator and filter) together with three-fourths of the solution from the battery, while the coarser, or over-size, is fed to a tube mill and ground to the proper fineness and discharged into the same storage settler, or agitator, acting as a storage settler, that the fines from the classifier were discharged into, where sands and slimes are brought together for succeeding treatment without further separation, thus permitting the reduction of the amount of solution before the ore is agitated to the proportion of one, two or three to one, by weight if so desired.

I have never found an ore in which the values could be extracted by leaching in two, four, six or eight days that as good or better extraction could not be made by proper and thorough agitation and aeration in the same number of hours. For that reason it is my practice to agitate sands and slimes together.

In placing the tube mill in the position above indicated in the mill I do not wish to be understood that all ores must be tube-milled, or finely ground, but that the great majority do I think is generally conceded.

The tube mill, or pebble mill, has within the past few years been brought to such a state of perfection that, in my opinion, it is far more economical to use a coarse screen in the battery and get the full efficiency of the stamps and finish with the tube mill. I prefer to use the tube mill because it has many advantages in fine grinding, such as dispensing with fine screens, which cause a continuous expense and which become clogged with finely ground wood pulp and other debris mixed with the ore, thereby reducing the capacity of the battery; the silex lining and flint pebbles do not cause a loss of cyanide as does the finely ground metal produced from the wearing parts of the iron or steel grinding machines. The wear of pebbles is much less in number of pounds and at a much less cost per pound than steel.

When sufficient ore and solution (say two to one) has accumulated in the agitator, acting as a storage settler, potassium cyanide is added to bring the solution up to a strength of from ½ lb. to 1 lb. per ton of solution for the average gold ores and not to exceed 1½ lb. per ton of solution for the average silver ore, and ½ lb. to 1 lb. of protective alkalinity (usually procured from calcium hydroxide), then the agitator is started and the ore pulp is agitated from two to eight hours for average gold ores and from eight to twenty-four hours for average silver ores. I have found many more ores that can be treated in two hours than those requiring twenty-four or more hours. During this period of agitation a second combination agitator and filter is used as a storage settler while agitator No. 1 is doing its agitating and filtering.

We have now only treated of that portion of the solutions which have remained in the agitator and are to be used for dissolving of the remainder of the precious metals which were not dissolved while passing through the grinding department. The surplus solution which has been taken off, if found to contain sufficient of the dissolved values and which has accumulated in the precipitation tank is mixed with zinc dust and passed through the zinc dust press, and having left its gold and silver values in the press is now termed "a barren solution." It is either run to the battery supply tank or to a barren solution tank above the agitator, where it is conveniently stored for the first wash of the tailings after the rich solutions have been filtered off from the agitator.

When the dissolving is completed in the agitator 80 per cent of the solution is drawn off through the filter cells and bottom canvas of the agitator and run to the precipitation tank above referred to, then the barren solution is drawn from its above-mentioned tank and introduced as a wash underneath the canvas in the agitator and through the filter cells and the agitator started and the whole mass put in a perfect mix. The agitator is then stopped and the filter cell cocks are opened and another 80 per cent of this solution, which is equal to 16 per cent of the original solution, is taken off. Then the same amount of fresh water that is to be finally lost in the tailings is added to the agitator on top of the settled charge and is drawn downward through the bottom canvas, displacing the remaining barren solution which was mixed with the remaining rich solution, or the charge is again mixed and discharged to the dewaterer.

By either procedure nearly the whole dissolved values have been recovered. This operation is made possible owing to the perfect homogeneous mix procured in this agitator. When the agitator shaft is stopped all the granular matter settles regardless of its mesh and covers the bottom canvas, thus forming a most perfect sand filter next to the canvas which will remain free from slimes sufficiently long to permit an ordinary suction pump to draw out the remaining solutions so that the remaining moisture in the pulp will not contain more than 20 per cent of moisture. (Fig. 1.)

The tailings are then ready to receive sufficient water (say 50 per cent) to allow them to be discharged from the agitator. They may be either discharged to the waste dump, if abundance

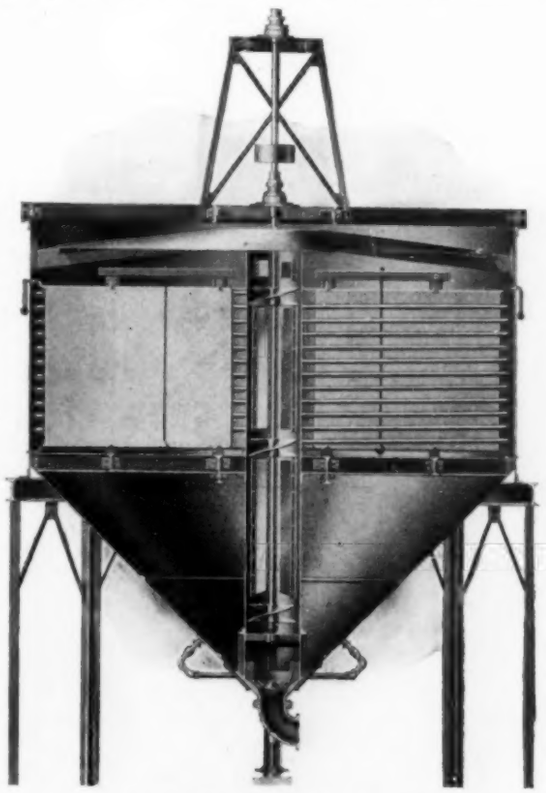


FIG. 1.—COMBINED AGITATOR AND FILTER.

of water is accessible, or if there is no objection to their flowing to the dump. In the event of greater economy being desirable owing to the lack of water, or owing to the fact that the tailings must be stacked in order to keep them from contaminating streams or flowing on to adjacent property, then they should be discharged into the dewatering device. (Fig. 2.)

After the addition of the wash water above mentioned, if it has been mixed with the remaining 4 per cent of the remaining pregnant solutions and barren solution, it should then be discharged to the dewaterer. The gold and silver filtered solutions are passed to the zinc precipitation tanks and pumped through the zinc dust press and returned to the battery storage tank or to the barren solution tank above the agitator. If there is not found enough alkalinity to protect the cyanide remaining in the solutions after passing through the zinc dust, which is often the case in heavy sulphide ores, caustic soda or hydroxid- of lime should be added to the battery solution tank from time to time in order to maintain a constant alkaline condition of the solution.

My experience has led me to adopt precipitation by zinc dust because the cost of operation is extremely low and besides almost complete precipitation is possible from a solution containing 1 lb. of cyanide or less per ton, thus making it possible to carry out the general scheme of using weak solutions all through the mill. The use of weak solutions lessens the liability of fouling, which occurs to a greater extent when strong solutions are used.

I do not believe any one who has had considerable experience in electrolytic work will presume to make the claim that a solution containing metals can be rendered barren of its metallic contents economically. On the other hand, there is a class of ore in which the solution becomes so fouled that it is extremely difficult, if not impossible, to successfully and economically precipitate the values on zinc. I have found that the introduction of electrolytic deposition in combination with zinc dust tends not only to regenerate the cyanide, but will deposit out the base metals, together with the gold and silver, leaving the solution in condition so that when passed through the zinc dust complete precipitation of the remaining gold and silver values is effected.

Combination Agitator and Filter

By referring to Fig. 1 it will not be necessary to describe the plain agitator, as the only difference between the two is the addition of the filter cells and the bottom canvas in order to convert the plain agitator to the so-called combination agitator and filter.

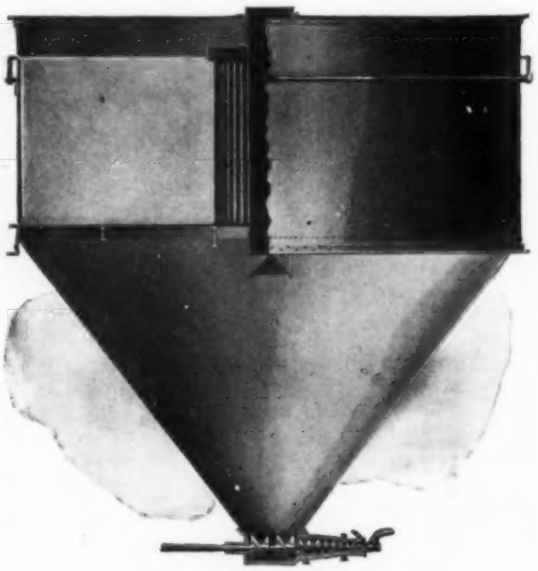


FIG. 2—DEWATERER.

The Hendryx combination agitator and filter consists essentially of four parts: a tank with converging bottom within which are centrally located an agitating pump, a series of filter cells, and a system of coils of pipes for heating the solution.

The tank is made of sheet steel and has a 45° bottom. A quick-acting discharge valve is located at the apex of the sloping bottom. Over the bottom of the tank is fastened a canvas sheet, which serves the purpose of evenly distributing the inflowing wash solution and allows the displacement of the solution remaining in the ground ore either upward or downward.

The agitating pump consists of a large well, within which is located a shaft carrying propellers. At the top of the well is a distributing and aerating apron which extends nearly to the sides of the tank.

The filter cells are located vertically beneath the distributing and aerating apron and consist of a wooden frame with iron tees and elbows. A piece of cocoa matting or wood strips fills the inside of the frame. The entire frame is enveloped in canvas, and the canvas and cocoa matting are quilted together at intervals. The filtered solution flows from a tee at a lower corner of the frame, while a tee at an upper corner serves the purpose of introducing air or wash solution to clean the cell.

The method of operating of the machine is somewhat as follows, although it may be varied to suit individual cases: Ore and solution in the proportion of about two of solution to one of ore, by weight, are accumulated either in the agitator or run in from storage until the tank is filled to nearly the bottom of the distributing and aerating apron. The agitating pump is then started and agitation continued until the soluble gold and silver have been put into solution. At the same time the solution is heated to about 90° Fahr. by means of hot water or steam circulating through the coils of heating pipe.

After the charge has been heated, agitated and aerated for a suitable time agitation is stopped, and the separate outlets to the filter cells are opened, allowing the solution to filter off by means of gravity. Owing to the perfect mixing in the agitator, when the power is shut off, a perfect classification of coarse and fine particles of ore takes place, so that the canvas in the bottom of the agitator has an ideal sand filter covering. This permits of the introduction of wash solution on top of the charge.

After filtering off approximately 70 per cent to 80 per cent of the original solution through the vertical cells and through the filter on the conical bottom, then by use of an ordinary suction pump, a downward displacement can be quickly accomplished. The downward displacement requires much less volume of water or displacing solution than the upward displacement and less time in the great majority of cases.

By the use of violent and rapid agitation and heat the slime is coagulated or curdled, so that it settles rapidly. Vacuum or pressure is not required on the filter cells, the solution filtering off by gravity alone.

Mechanical agitation is the only method of agitation that puts the charge in this condition, and the only method which gives the shortest time for dissolution, as the mechanical agitator with the same horse-power will circulate the ore pulp ten times as rapidly as the air agitator.

Air agitation also causes emulsification of the slime, thereby retarding filtration, prohibiting settling of the charge for decantation, introducing an excess of air containing carbon dioxide, which is a cyanicide, also permits of the introduction of oil from the use of a compressor, which still further emulsifies the solution and forms hydrocarbons, another source of cyanide loss.

A machine of this character capable of agitating and filtering 100 tons of average gold ores per day requires only 6 hp. With this machine the solutions can be readily changed; that is, 80 per cent or less as desired of the first solution can be taken off and a fresh solution added which, it has been found in the treatment of some of the silver ores, decreases the time of treatment as much as 25 per cent.

Storage or Tailings Dewaterer.

This dewaterer is designed to remove the water or solution from concentrates or tailings and deliver the same in such condition that they can be conveyed away on a belt or in a car and stacked. The solution or water is recovered in a clear condition and returned for further use.

It is also used in the case where only one agitator is installed to thicken slimes and to collect and hold the ground ore until such time as a sufficient tonnage has accumulated to make a charge for the agitator, when it is discharged. The excess solution to the dewaterer from the crushing department having been recovered and returned to the grinding department.

Where the water supply is limited, with this dewaterer the mill capacity will be more than doubled besides making a closer saving of the values and preventing the loss of cyanide and chemicals in the discharged tailings.

This machine without the use of vacuum or pressure will de-

water 200-mesh tailings down to from 12 per cent to 20 per cent moisture.

This apparatus consists of a round tank with a conical bottom. Within the tank are placed vertical filter cells for the purpose of clarifying the solution or water. At the bottom of the cone is an outlet within which is a screw connected to a shaft, which is revolved.

The tailings as they settle are removed from the conical bottom and forced through the outlet by the revolving screw in a continuous stream. The screw is run at a very slow speed, hence there is very little wear on any of the working parts.

This machine is used after the combination agitator and filter in order to recover the wash solution and fresh-water wash, thereby making the highest possible saving of chemicals and also shortening the time in the combination agitator and filter, and thereby increasing its capacity.

New York City.

The Analysis of Aluminium Alloys.

BY THORN SMITH.

The methods for the analysis of aluminium alloys as given in many textbooks are crude and inapplicable, so much so that one is very apt to conclude that the writers have had no experience and are merely describing methods which they think are applicable. Much annoyance and loss of time follow when their application is attempted by the chemist who but seldom has occasion to use them. Time and again the estimation of aluminium by precipitation with ammonia has been proven worthless in the hands of the ordinary chemist, yet textbook writers persist in giving the method first place; in fact, it is generally the only one given. To suggest that 1 gram of aluminium can be weighed out, dissolved in caustic soda, made acid wth hydrochloric, and precipitated as hydroxide with ammonia, then filtered, washed and ignited, is beyond reason and, to speak plainly, ridiculous. So much has been written on this method of procedure as usually carried out that further comment at this time is unnecessary.

It is sufficient to say that at the present time there is no method for the direct estimation of aluminium in alloys high in this element and that the amount present as obtained by difference is the best that can be done. Therefore, if even the approximate percentage be desired, it is essential that care be taken in estimating the other elements in the alloy and that the chemist have a full understanding of their solubilities and interferences. Considerable latitude must be allowed as the various alloys are radically different and the procedure may be greatly shortened in many cases. New methods are not common, but the older methods can in many instances be shortened and presented in a new light. The present article is written to help those who but seldom have use for such methods and at the same time to furnish thinking points for those who have occasion to analyze these alloys frequently.

Analysis of Commercial Aluminiums.

The two impurities present are silicon and iron. Neither of these present any difficulty if ordinary precautions are taken.

Silicon.—Dissolve 1 gram of the drillings in 30 cc of the following mixture: Nitric, 1 part; hydrochloric, 3 parts; sulphuric, 2 parts, and water, 4 parts. A 4-in. porcelain casserole well covered is the best for dissolving the drillings. When violent action ceases, add an additional 5 cc of sulphuric acid and take rapidly down to white fumes. This is quickly done when the solution is nearly evaporated by rapidly rotating the casserole over an open burner. Taking down to complete dryness or evaporating off the excess sulphuric acid is unnecessary. Allow to cool and add 50 cc of cold water, stir, and boil until the sulphates are in solution, a point easily noted. Filter *at once*, washing with hot water, containing a little HCl, several times and finally with hot water alone until the filtrate is free from chlorides. Ignite the moist precipitate in an unweighed platinum crucible and after the paper is destroyed blast for at least five minutes. Cool and weigh. To the crucible add 1 drop of sulphuric acid and 10 drops of hydrofluoric acid. Evaporate on the hot plate to the drop of sulphuric acid, then heat cautiously over a burner until the sulphuric is expelled. Blast five minutes, cool and weigh. Factor 0.4693 equals silicon. There should be no appreciable residue left in the crucible.

Iron.—Contrary to the usual manner of estimating the iron by reducing the silicon filtrate and titrating with permanganate a separate portion of from 2 grams to 5 grams is weighed into a No. 3 beaker and from 10 cc to 20 cc of 1 to 1 hydrochloric acid cautiously added. When in solution boil a few minutes and add a drop or two of stannous chloride to reduce any iron that may have been oxidized. From this point titrate with normal twentieth, or weaker, permanganate solution, using a titrating mixture and following some definite method of procedure, preferably that used by the United States Steel Corporation, as described in the February, 1909, number of this journal (Vol. VII, page 65). It is important that the permanganate be standardized in the same manner. Another point of no less importance is the titration of an iron-free blank and deducting this figure from all titrations even in standardizing. With these precautions the results are accurate, providing titanium is absent.

Aluminium Alloys.

The one most commonly in use contains from 5 per cent to 10 per cent of copper in addition to the impurities mentioned above.

The silicon is estimated in the same manner. For the iron proceed as above until solution is effected, dilute slightly with water, filter and wash two or three times with hot water. Heat to boiling, reduce with tin solution and titrate as above. The little copper that goes into solution does no harm. The iron usually present in the alloy is so small and the solution so strongly acid that the amount held up by the copper is negligible.

Copper.—A 1-gram or 2-gram weight is taken, depending on the amount of copper present. Place in a No. 3 beaker, add 5 grams or more of sodium hydroxide and enough water to cover. When action ceases add 200 cc of hot water and boil five minutes or more. Filter and wash once with hot water. Dissolve the precipitate on the filter with 1 to 1 nitric acid, allowing the filtrate to run into the beaker in which the alloy was dissolved. Wash the paper three or four times, then pour on ammonia until the solution is deep blue. If the filter paper is colored blue, due to copper, wash again. To the blue solution, which should not exceed 125 cc, add 2 cc each of sulphuric and nitric acids. This amount should make the solution acid. Electrolyze.

Columns might be written on the electrolytic method of determining copper, so confusing are the textbook methods. Suffice to say that given a copper solution which may contain any or all of iron, aluminium, zinc and manganese the plating process is simple. Whatever form of cathode is employed the main precaution to be observed is the avoidance of too heavy a current, which will cause sponging or blackening of the deposited metal. The voltage may be ignored providing it is not too low; in other words, a 125-volt, direct-current lighting circuit may be used successfully. The rate at which the electrolyte gases is the best measure of the amount of current to use and a little experience is sufficient to guide one.

While for first-class work the electrolytic method is to be preferred, there are times when the chemist may prefer to use the cyanide method. To this there is no objection providing a strict method of procedure be followed. In dissolving the copper precipitate use 10 cc of 1 to 1 nitric acid in every case. Wash the paper thoroughly and boil off the nitrous fumes. Cool and add 10 cc of ammonia. Make up to a definite volume, preferably 100 cc and titrate with KCN, 1 cc of which equals 2/10 per cent of copper. In standardizing the cyanide the metallic copper should be dissolved in 5 cc of nitric acid and

made alkaline with 10 cc of ammonia. The volume should be the same as before. It is best to titrate to a faint pink color rather than to the complete discharge of the blue color. The addition of the cyanide should be very slow as the end point is approached. The presence of iron, etc., in the solution does no harm and it is a waste of time to remove them. The main precaution is that the operator is sure that he has all of the copper in the solution which he is titrating that originally existed in the alloy. The addition of hydrogen sulphide to the solution before separating the metallic copper by filtration is advisable. If much sulphide is thrown down it is well to use a little bromine water with the 1 to 1 nitric acid. The same amounts of acid and ammonia should be used in every titration and in titrating the operator should not go too far. Much time may be saved by having a standard solution of copper on hand made by dissolving 2 grams of copper foil in as little nitric acid as possible and diluting to 1 liter. Determine the copper in 25 cc of this solution by the electrolytic method. In using this solution as a standard do not omit the addition of 5 cc of nitric acid.

When the alloy contains *zinc* in addition to the iron and silicon proceed as before for these elements. For the zinc dissolve 0.5 gram to 1 gram, or more, of the alloy in dilute hydrochloric acid, boil a few minutes and cool. Make up to a volume of about 100 cc, add three drops of methyl orange, then sodium bicarbonate, until the solution is just straw color. Now add 20 per cent formic acid, drop by drop, until the solution again shows acid, then add five drops in excess. Dilute to 200 cc or 300 cc with hot water and heat to very near boiling. To this hot solution run in hydrogen sulphide until precipitation is complete. Allow the precipitate of zinc sulphide to settle, filter and wash by decantation three times, using hot water. Wash the paper to remove aluminium, then throw back into the beaker in which the precipitation was made; add 50 cc of hot water and 10 cc of hydrochloric acid. Boil until the zinc sulphide is completely dissolved. Filter, washing with hot water containing a few drops of hydrochloric acid. Boil off the hydrogen sulphide and add a few drops of bromine water to oxidize any possible iron. Cool slightly and add ammonia in excess. Boil and filter. If the precipitate is more than a trace reprecipitate. Now make the solution slightly acid with hydrochloric and add a *large* excess of sodium or ammonium phosphate. Make just alkaline to litmus paper with ammonia, then add 1 cc of strong acetic acid. Warm until the zinc phosphate is crystalline in appearance, which requires but a few minutes. Allow to settle, filter and wash by decantation several times with hot water before adding the precipitate to the filter. Dry the paper and precipitate, detach the latter and burn the paper in a porcelain crucible. Then add the precipitate and ignite very gently at first, increasing the heat gradually and finally heat to bright redness for five minutes. Factor 0.4291 equals zinc.

This is at present the most accurate method we have for the determination of zinc. Too much emphasis cannot be laid on the use of an excess of the phosphate salt.

Where strict accuracy is not desired titrating with ferrocyanide is very satisfactory, the results depending on the experience of the analyst with the method and the strict observance of uniformity in all operations.

When copper is present with the zinc it is better to dissolve a weighed portion in sodium hydroxide, dilute to 200 cc and heat to boiling. Precipitate the copper and zinc together as sulphides. The presence of the copper permits the filtering of the mixed sulphides without the usual running through the filter paper. Wash once with water containing hydrogen sulphide to remove the greater part of the aluminium. Place filter and contents back into the beaker in which the precipitation was made; add 50 cc of hot water and 10 cc of hydrochloric acid. Boil to dissolve the zinc sulphide and filter. Wash with water containing a few drops of hydrochloric acid and a little hydrogen sulphide. Boil off the hydrogen sulphide, add bromine water and proceed as before, making a double separation with ammonia if the aluminium is in much evidence. From this point proceed as above.

The estimation of *manganese* is frequently desirable and there is no method to equal the "Bismuthate." A weighed portion up to 5 grams is dissolved in sodium hydroxide, diluted, boiled and filtered. The manganese is on the filter. Dissolve by pouring on 50 cc of nitric acid, sp. gr. 1.135, containing a few drops of sulphurous acid. From this point proceed with the bismuthate method.

Tin, antimony, lead, nickel, titanium and even other elements may be found at times, depending on the source of the alloy or the use to which it is to be put. None of these present material difficulty and can be estimated by well-known methods.

In conclusion, the chief causes of poor results are: The impossibility of securing an even approximate result by estimating aluminium in the usual manner. This is increased by the use of glassware in precipitating.

The attempt to estimate iron on too small a weight of the material. The attempt to precipitate and filter zinc from an alkaline solution in place of the newer and better method using formic acid.

The failure to adhere strictly to methods as given and making no departure until convinced by repeated experiment that harm will not be done.

Note.—The reader is referred to the following publications for more detailed descriptions of methods mentioned in this article: *Bulletin* 304 U. S. Geolog. Survey; *Journal* of the American Chemical Society, volume 26, page 4, and volume 20, page 262. Blair's "Chemical Analysis of Iron" for the bismuthate method.

Detroit, Mich.

Formulas for Pinch Phenomenon.

By Carl Hering.

The internal electromagnetic force, which acts to contract the cross-section of a conductor, due to the current, and which is now known as the pinch phenomenon, occasionally enters into calculations, especially in connection with electric furnaces in which there is a liquid resister in which such relatively small forces can act and must therefore be considered. In such cases it is desirable to have the formulas for calculating it in convenient form for direct use, as errors often arise in reducing a formula from one kind of units to another. The purpose of the present note is to give these formulas in their different forms.

The original formula deduced by Dr. Northrup[1] was intended for a particular purpose and is not in the most convenient form for direct application to furnaces. It is in pounds per square centimeter, a unit of pressure seldom used, and it contains as two factors the cross-section and the current density, hence the latter depends on the former.

Omitting the factor n, which is unity in the general case, this original formula is as follows:

$$P = \frac{S\,C^2}{44{,}479{,}100} \qquad (1)$$

P = pressure in pounds per square centimeter.

S = section square centimeter.

C = current density in amperes per square centimeter.

It applies strictly only to cylindrical conductors. P is the pressure at the center. The constant is a purely mathematical one and involves no empirical determinations.

In the form thus given the formula brings out the interesting fact that the pressure increases as the *square* of the current density. This, however, is somewhat deceptive in that when the current density is increased by diminishing the section then

[1] "Some Newly Observed Manifestations of Forces in the Interior of an Electric Conductor," *Physical Review*, Vol. XXIV, No. 6, June, 1907, p. 474.

the pressure will not increase in that proportion because the other factor S will be decreased also. It is for this reason that the writer does not favor the use of such interdependent quantities in formulas when it is just as easy to use entirely independent ones.

By reducing it to English units and eliminating the current density, hence using only independent factors, the formula reduces to

$$P = 0.000,000,022,482,5 \frac{C^2}{S} \qquad (2)$$

P = pounds per square inch.
S = square inches.
C = current in amperes.

To reduce this awkward constant to a simpler one the cross-section may be represented in square mils and the formula will then reduce to

$$P = 0.022,482,5 \frac{C^2}{S} \qquad (3)$$

P = pounds per square inch.
S = square mils.
C = amperes.

To reduce it to the metric system it is preferable to express the cross-section in square millimeters instead of square centimeters, as the constant is then less unwieldy. The formula then becomes

$$P = 0.001,019,79 \frac{C^2}{S} \qquad (4)$$

P = grams per square centimeter.
S = square millimeters.
C = amperes.

The reason for the awkwardly small constant is that the formula involves the square of the current, which is generally a numerically very large quantity, and as the pressures are not very large numerically the constant must necessarily be very small. Calculations are therefore simplified and the constants more easily remembered by expressing the large currents C in kilo-amperes (1000 amp) instead of amperes. Formula (2) in inches and pounds then becomes

$$P = 0.022,482,5 \frac{C^2}{S} = \text{approximately } \frac{9\,C^2}{400\,S} \qquad (5)$$

P = pounds per square inch.
S = square inches.
C = kilo-amperes (1000 amp)

and the one in the metric system, S now being in square centimeters instead of square millimeters, becomes

$$P = 10.197,9 \frac{C^2}{S} = \text{approximately } \frac{10\,C^2}{S} \qquad (6)$$

P = grams per square centimeter.
S = square centimeters.
C = kilo-amperes (1000 amp).

The two approximate formulas (the errors are less than 2 per cent) are now quite simple and easily remembered.

In the c.g.s. system the formula is simply

$$P = \frac{C^2}{S} \qquad (7)$$

P = dynes per square centimeter; S = square centimeters; C = current in c.g.s. units (10 amp).

Attention is here called to the fact pointed out by the writer some years ago[2] that complete rupture of a conductor can take place long before this pinch pressure has reached the hydraulic pressure at the bottom of a trough. This is due to the fact that after a certain amount of shrinking of a conductor an unstable state is reached after which further shrinkage takes place automatically and by rapidly increasing amounts, thereby causing complete rupture. This should be taken into consideration in applying the above formulas to such cases.

Philadelphia, Pa.

[2] "The Working Limit in Electrical Furnaces Due to the 'Pinch' Phenomena." *Trans.* Am. Electrochem. Soc., Vol. XV, 1909, p. 255.

Annealing of Copper and Diseases of Copper.

By F. JOHNSON, M.Sc.

The annealing of copper is designed to remove hardness and brittleness conferred on the metal either by cold working or by the repetition of minute stresses. Softness and toughness, which enable the metal to undergo further stresses without being dangerously embrittled, are regained.

There are many conditions to be observed during annealing, chief of which are as follows:

(1) *Correct temperature.*
(2) *Neutral or oxidizing atmosphere. An atmosphere of a reducing nature must be strictly avoided.*
(3) *Absence of foreign material from annealing furnace.*
(4) *Uniformity of heating.*
(5) *Duration of annealing.*
(6) *Method of Cooling.*

The observance of all these conditions is profoundly affected by the size, thickness and form of material which is to be annealed. Generally speaking, the thinner the material the more susceptible it is to injury by abnormal treatment.

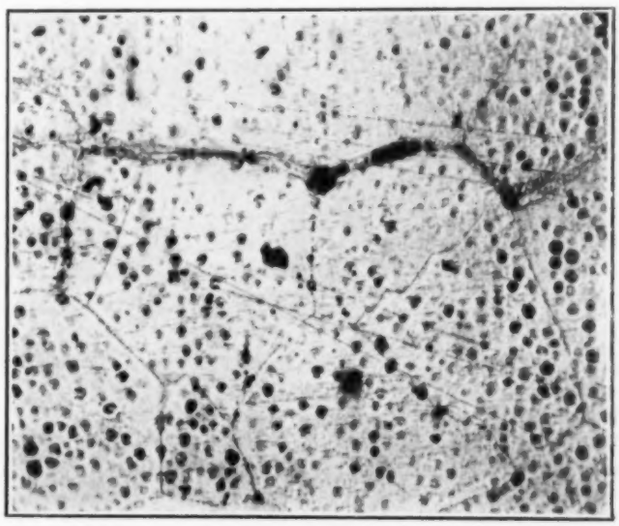


FIG. 1—WROUGHT COPPER. "BURNT" IN ANNEALING. MAGNIFICATION 100 DIAM.

(1) *Temperature.*—A bright red heat, which corresponds to about 900° C., is not too high for the purest copper. Impure copper of the refined type, if capable of being satisfactorily worked, should also easily stand this temperature.

The danger in raising copper to too high a temperature during annealing (1) lies in the occurrence of incipient fusion, but (2) is much intensified if gases of a reducing nature be present. If the annealing is in the nature of a reheating preparatory to hot working, a bright-red heat may not be exceeded, for to work copper at a temperature above 1000° C. is suicidal. In the neighborhood of this temperature copper is extremely brittle and rotten, possessing neither strength nor toughness, the reason being, possibly, that the cuprous oxide eutectic assumes a pasty condition. At any rate it becomes brittle and unmalleable with an alarming abruptness, quite incommensurate with the small rise above the forging temperature.

If the annealing temperature be too low then crystalline readjustment is not complete, i. e., the hardness and brittleness due to stressing are not sufficiently removed and the material is liable to rupture if again submitted to cold work. The presence of cuprous oxide ("oxidules")* tends probably to promote recrystallization at lower temperatures than in the case of oxygen-free (deoxidized) copper.

Micrograph No. 1 shows the structure of a piece of cold-

* The author: *Journal Inst. of Metals*, Vol. IV, No. 2.

wrought copper which has been allowed to attain to too high a temperature in the annealing process, with the result that incipient fusion has been accompanied by intracrystalline oxidation causing oxidized boundaries and intracrystalline pitting.

2. *Atmosphere of Furnace.*—This should be carefully watched, as, if gases of a reducing nature, such as H, CO, CH_4, etc., all of which may be present in the products of imperfect combustion of carbonaceous fuel, are allowed to pass over copper at a red heat, or even at a lower temperature, they remove the all-important oxygen, rendering the metal porous, brittle, weak, destitute of all the properties which good copper possesses and,

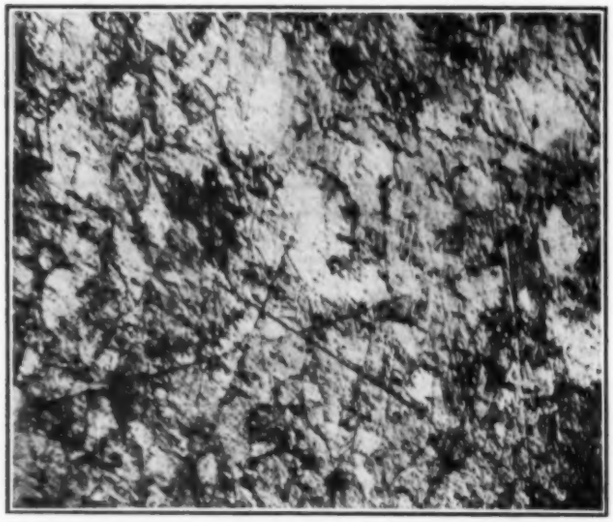


FIG. 2—WROUGHT COPPER (ARSENICAL) NORMALLY ANNEALED.

finally, amenable to no remedy short of remelting. This is commonly known as "gassing," the copper being the more sensitive to the action the higher the temperature.

3. *Foreign Material in Furnace.*—There should be no sharp grit, nor other foreign material at all on the floor of the annealing furnace, which must be kept perfectly clean. Apart from the mechanical action of such material in scratching or grooving copper, causing initial flaws which may engender failure in final tests, there is the possibility of the copper absorbing zinc from articles of brass. Brass articles should never be annealed in the same furnace and at the same time as copper articles.

Just as mercury will amalgamate with copper at ordinary temperatures, on account of the remarkable molecular fluidity and absence of viscosity of the former, so will zinc suffer a transference of its molecules to take place from the brass which contains it to copper in contact with it at a high temperature.*

So it is never advisable to calcine, in a muffle used for annealing copper, any refuse or sweepings, especially such as contain particles of brass. Otherwise the copper may take up defects, which, causing subsequent erratic behavior, may help to still further thicken that shroud of mystery which all too completely envelops the metal even now.

4. *Uniformity of Heating.*—This is of extreme importance. There are, perhaps, few works where wrough-copper material of large dimensions is annealed uniformly all over. The practice, for instance, of annealing large copper pipes over a smith's fire, bit by bit, is quite common. Where intricate shapes and sharp bends are being dealt with it is by this method impossible to insure the desirable uniformity of heating. Where furnaces are used the closed type is preferable to the open type, for, in addition to insuring uniformity of heating, all injurious gaseous products of combustion of the fuel are kept from contact with the heated metal. But whatever type of furnace is used it should be the invariable rule of the furnace man to see that every inch of metal has come to a red heat, even if it is necessary to reverse the material before withdrawal.

If this is not done the metal will vary considerably in its properties, and if it is to be put into service where, as in the case of a steam pipe, it will have to undergo the inevitable stresses due to alternate expansions and contractions, and to vibrations, it is quite certain that those parts which have been heated to the highest temperature and are, in consequence, softest, will yield most to the stresses. This localization of strains results in premature cracking and rupture of an otherwise satisfactory pipe.

5. *Duration of Annealing.*—There is no advantage in prolonging the heating beyond the attainment of a ***uniform*** red heat. The great danger is that the heating may be sufficient at one part and, while waiting for the remaining parts to come to the same temperature, the first part may be overheated or even "burnt."

Overheated metal, or metal which has been kept too long in the furnace, is softer than metal which has been just annealed and then withdrawn, as shown by the following figures:

Just Annealed.	*Over Annealed.*
Tensile Str. tons per sq. in.	*Tensile Str. tons per sq. in.*
15.12	14.24
See Micrograph No. 2.	See Micrograph No. 3.

Prolonged annealing induces crystalline growth and has the invariable effect of lowering the tensile strength and possibly the percentage elongation, although the latter may not be altered at all if the temperature has not exceeded a dull-red heat. Under the microscope it will be seen that the crystals have increased in size, small crystals being absorbed by larger ones.

6. *Method of Cooling.*—Copper may be quenched from a bright-red heat or cooled slowly. Quenching (1) has the effect of removing scale and retaining it in the bosh until cleaning

FIG. 3—WROUGHT COPPER (ARSENICAL) OVER-ANNEALED.

out becomes necessary. This prevents the scale, a material of considerable value, from being trodden about on the floor and lost. (2) Softens the metal and toughens it against alternating stresses. The following figures are from a paper read by Mr. J. M. Allen, January, 1909, before the N. E. C. Inst. of Engineers and Shipbuilders. The test pieces were all taken from the same sheet and were bent backward and forward by hand through 90° and over a curve of 1¼-in. radius until fractured. The figures given denote the mean number of alternations endured in two such tests:

* *The Analyst*, Vol. XXX, No. 357, Dec., 1905, L. Archbutt; also, *Journal Inst. of Metals*, Vol. III, No. 1, Bengough & Hill, p. 45.

	Sheet No.	Mean No. of times bent.	Tube No.	Mean No. of times bent.
Untreated Strips ...	S 1	36	T 1	61.5
Heated to dull red and slowly cooled.	S 10	38	T 2	92
Heated to dull red and quenched.....	S 3	39.5	T 3	99.5
Heated to bright red and slowly cooled.	S 4	43	T 4	93
Heated to bright red and quenched.....	S 11	57.5	T 11	113

Judging from these figures it would seem to be the best practice in the technique of annealing to heat to a bright-red heat and quench rapidly, thereby gaining a maximum toughness

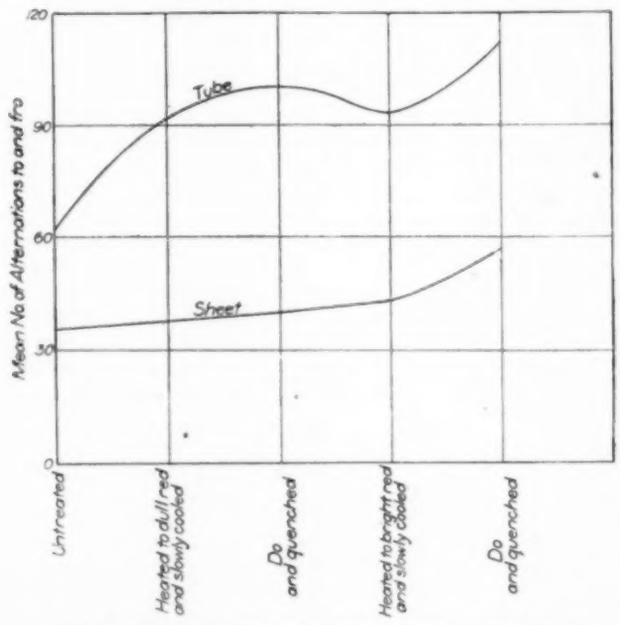


FIG. 4—EFFECT OF HEAT TREATMENT ON BEHAVIOR UNDER ALTERNATING STRESSES.

with no lowering of tensile strength or percentage elongation. The advantage of quenching seems to be the more marked (relative to slow cooling) where the copper has more lead in its composition. (As in the case of the sheet [S], see curves Fig. 4.)

Diseases of Copper.

Corrosion.

Chemical Action.—This may occur in many ways and its continuance or retardation will depend on (a) whether the product of the solvent action is protective or contributory to further wasting, (b) whether it is continuously removed so soon as formed or allowed to remain undisturbed in its protective or in its corrosive action.

In case (a) the sheet copper used for covering domes of buildings may be instanced. After exposure for some time to a marine atmosphere a coating of cupric chloride or oxychloride is formed which confers a distinct ornamental effect and is at once a protection against further atmospheric attack and an embellishment. Again, articles of copper exposed to similar action, either of sea water or other corrosive liquid, may, owing to the fact that they are in constant or intermittent motion, be continuously exposing clean surfaces to further corrosion and thus wasting gradually away.

Articles of copper exposed to a corrosive liquid and permanently removed therefrom, if not very carefully cleaned and dried, will harbor the corrosive agent in minute superficial pits or grooves and corrosion will proceed, more slowly than before and more erratically, but quite perceptibly.

Rhead ("The Corrosion of Copper and Brass," J. Inst. of Metals, Vol. 11, No. 2, p. 87) has shown that copper hardened by mechanical work is more susceptible to corrosion in acid and saline solutions than copper softened (by annealing) under similar conditions.

Sir G. Muntz (discussion on "The Failure of Non-Ferrous Metals," T. V. Hughes, Inst. of Metals, January, 1910) instanced alkaline water in boilers as the cause of intense pitting and riddling of boiler tubes.

Bengough (ibid.) stated that salt water in boilers causes corrosion on the water side of the tubes, whereas fresh water is inert, corrosion in the latter case occurring only on the fire side of the tubes. He thinks that particles of slag in the metal may form "loci" where corrosion will be first fostered, but it has yet to be proved that copper or brass tubes contain particles of slag. Even admitting the possibility of their presence in copper billets or brass castings—a possibility which is extremely remote—they would prevent the forged billet or the casting from surviving the drawing process.

Handscomb ("Characteristics of Copper under Various Conditions," Inst. Marine Engrs., September, 1909) attributes pitting in steam pipes to the ammoniacal nature of patent compounds used as heat-non-conducting coverings. In all cases of chemical corrosion the action is hastened and intensified in proportion to the temperature; the higher the temperature the more rapid the corrosion.

Galvanic Action and *Decay* have been very fully dealt with by Milton and Larke in Proc. Inst. Civil Engrs., 1903.

Gaseous Action in Course of Manufacture.—The effects of "burning," overheating and "gassing" have already been dealt with under annealing and it is extremely unlikely that wrought copper which has been seriously injured during the processes of manufacture will be allowed to issue from the works of any reputable firm of manufacturers.

Careless treatment of copper steam pipes over the coppersmith's fire may cause "gassing" of the exterior of the pipe. If only the merest superficial layer is deoxidized trouble is being brewed, for incipient cracking, even if well-nigh imperceptible, is unspeakably dangerous, since the propagation of larger cracks throughout the thickness of the pipe is thus freely encouraged and the life of the pipe assuredly shortened.

Gaseous Action (B) in Service.—Hot gases from the combustion of fuel may consist of inert gases, such as N and CO_2 and corrosive gases, such as Cl, SH_2, SO_2, COH, etc. Chlorine, sulphuretted hydrogen and sulphur dioxide must, if present in the furnace gases of a locomotive boiler, cause wastage of a serious character. They would act in the formation of a superficial layer of scale, free oxygen being probably necessary to complete the chemical equation of which an oxychloride or sulphate would be the probable product. Such a salt would be friable and non-adherent, easily shaken off by vibration or by the abrading action of cinders and dust, thus constantly exposing a fresh surface to further corrosion.

Carbon monoxide would only act on copper at a high temperature (say, incipient red) and then it would have a more than superficial effect; it would penetrate into the metal, reduce cuprous oxide present, and leave in its wake a weakened structure. It is almost certain, however, that boiler *tubes* at all events, even at the firebox end, never reach so high a temperature as to admit of the insidious inroads of reducing gases.

The abrading action of *cinders* and the "burning" action of *flame* may be noted under this head, since they are inseparable from the onward sweep of furnace gases caused by the fierce draught. Webb (Inst. C. E., 1903) showed how the red-hot cinders and ash were caused to impinge on the lower half of the boiler tube, where the latter were held into the firebox tubeplate at the ferrule, and were deflected thence in an upward direction, meeting the top half of the tube a few inches further on. The great majority of failures occurred in these two places. Cinders and flame are responsible for much of the wastage of stay-heads and "scoring" of firebox plates.

Here, too, may be mentioned the theory of local overheating (due to oil in the feed water adhering to parts of the firebox and causing a retardation of the conduction of heat to the water). Such spots would, it is thought, be subject to intense local heating, causing, on the fire side of the plate, incipient or partial fusion and subsequent weakness.

The greatest trouble experienced with circulating *water pipes* of copper is internal pitting of a local character. This has been traced in some instances to the liberation of air from the circulating water. Sea water is known to contain air in solution and if the conditions are such that the supply of water to the pipes is inconstant, they will not always be quite full, with the result that free air will be liberated, and rising to the "roof," as it were, of pipes in a horizontal position, will attack the copper under the necessarily moist, saline conditions obtaining, often in an erratic manner, causing the pipe to scale. Scale thus formed will be shaken off by vibration, and "pitting" will proceed until at length a hole is formed right through.

Ensuring that the pipes are always quite full of water has been the means of preventing further corrosion of this nature.

The presence of impurities in wrought copper may be considered as helpful, if anything, in enabling the copper to resist corrosion. Certainly the purest electrolytic copper is just as susceptible to corrosive influences as commercial "tough" copper, and possibly more so.

Swansea Technical College,
Swansea, England.

The Work of the Chemical Laboratories of the Bureau of Mines.

Dr. J. K. Clement, the physicist of the Bureau of Mines, presented an interesting paper at the recent Minneapolis meeting of the American Chemical Society on the chemical laboratory work of the Bureau of Mines.

This work is divided among a number of separate laboratories, each carrying on its own lines of work under the direction of its own chief, the whole forming a group of more or less independent units. The relation of the work of the several chemical laboratories to that of the other departments of the bureau varies with individual cases. In general, however, the problems of the chemists are closely connected with those of the mining and mechanical engineers.

The fuel testing laboratory is occupied mainly with the analysis and calorimetric testing of fuels, including coal, coke, lignite and peat. In addition to analyzing samples of all fuels used in the boiler and gas producer tests of the bureau, ultimate analyses and calorific value determinations are made on mine samples of coal collected at various places. The data on these latter tests are of value in establishing the composition and heating value of the coals in connection with the classification of the coal fields of the United States.

In addition to the laboratory at Pittsburgh, there is located in Washington, D. C., a laboratory in which are tested samples of coal, delivered to the various buildings, arsenals, navy yards and military posts within the District of Columbia and in various parts of the country, and of the coal purchased by the Panama Railroad.

Fusibilty and Clinkering of Coal Ash.—In the use of coal under steam boilers, the property next in importance to its calorific value is perhaps the fusibility of its ash. Indeed, some coals, which have a high heating value, are worthless for making steam on account of their tendency to clinker and adhere to the grate bars. The relation between the fusibility and clinkering properties of coal ash and its chemical and mineralogical composition is now being investigated.

It is interesting to note that TiO_2 was found in all the clinkers examined in amounts varying from 1 per cent to 3 per cent. Determinations of the fusion point of various ashes give values ranging from 1150° C. to 1400° C.

Chemistry of Petroleum Technology.—The bureau is making a study of the commercial bodies contained in the crude petroleums of the United States; of the methods for their separation and purification and of their economic uses. The California fields, because of their showing at this time the greater promise of a large and continued production, their promixity to naval stations and the peculiar adaptability of their product as a maritime and a locomotive fuel, have been selected for first study. Besides the determination of the properties and uses of the various products of the petroleums of the country, an investigation is being made of the processes of distillation and of the methods of refining.

Combustion Investigations.—The processes of combustion in the boiler furnaces are being investigated in a furnace specially designed for the purpose. By taking simultaneous samples of the combustion gases, the progress of the reactions may be followed and the time or space necessary for the complete combustion of various coals and under varying conditions of operation may be determined.

The process of producer gas formation is being studied from a physico-chemical standpoint, and an attempt will be made to apply on a commercial scale the results of laboratory experiments on the rate of formation of carbon monoxide and water gas.

The Composition of Coal.—Our scientific knowledge of the chemical character of coal is limited almost entirely to its chemical analysis and its adaptation to certain industrial operations. The object of one of the investigations of the bureau is the isolation and identification of some of the constituents of coal. By the use of inert solvents it has been found possible to extract as much as 35 per cent of the original coal. A number of different substances have been isolated and the analysis and molecular weights of some of these substances have been determined. In a few cases it is believed that the materials are practically pure substances.

The Volatile Matter of Coal.—The quantity and composition of the gases evolved from various coals, when heated to temperatures of from 400° to 1000° C., have been determined. In the experiments which are now in progress particular attention will be given to the influence of the rate of heating on the character of the gases produced; to the initial composition of the gases at the instant of liberation and to the thermal decomposition of these gases during passage over heated surfaces.

Weathering and Deterioration of Coal.—In co-operation with the Navy Department, the Panama Railroad Company and the University of Michigan, the Bureau is conducting an extensive series of tests on the deterioration of various coals in storage both in the open air and when submerged in fresh water and sea water.

The Accumulation of Gas from Coal.—The quantity and rate of formation of inflammable gas from freshly mined coal, at ordinary temperatures, and the rate of absorption of oxygen by the coal have been determined.

The Spantaneous Combustion of Coal is being investigated by the Bureau. Statistical information will be combined with the results obtained in the laboratory.

The Burning of Coal in Mines under a Diminished Supply of Oxygen.—The factors governing the propagation or extinguishing of fires in mines are being investigated. Chief among these are variations in temperature and in the oxygen content of the surrounding atmosphere.

Examination of Mine Gases.—Examination is made of samples from normal mine air, from the after-damp following explosions, from stagnant areas and from burning areas during mine fires. Particular attention has been given to the detection of small amounts of carbon monoxide. By analyzing samples during the progress of mine fires, the chemist has assisted in combating fires. The effect of variations of barometric pressure on the exudation of methane and the influence of carbon dioxide on the explosibility of mine gases are being investigated.

The Chemistry of Explosives.—Chemical analyses are made of all explosives submitted to the Bureau for test, of the products of combustion of explosives and of electric detonators, blasting caps and fuses. All explosives, blasting caps, electric detonators and fuses purchased by the Isthmian Canal Commission are inspected by representatives of the Bureau, and all shipments of such explosives are sampled and analyzed.

Coal Dust Explosions.—The two greatest sources of danger encountered in mining operations are the explosive gases given off by the coal, and the finely divided coal dust which exists throughout most coal mines. The first danger can be overcome by increasing the ventilation in the mines. Unfortunately, this increases the danger from the coal dust by the removal of its moisture.

A laboratory method has been devised to test the inflammable character of samples of coal dust, and to classify them according to their inflammability. This method is based on determining the amount of combustion which takes place when clouds of dust of the same density are ignited under the same conditions, the amount of combustion being determined by the pressure developed within the explosion vessel. In this way it is possible to obtain results on any one sample of coal dust which agree to 3-5 per cent of the total pressure developed.

One of the proposed means of lessening the inflammable character of coal dust is to add a non-inflammable dust. The laboratory method used to investigate the inflammability of coal dust has been extended to various percentages of coal dust and finely ground shale in order to determine to what extent the combustion is limited by the presence of the inert dust. The experiments indicate that a marked diminution of pressure is not obtained until about 25 per cent of inert dust is added, the pressure then falling off rapidly with a further increase in the amount of shale dust added.

Western Chemists and Metallurgists Disband.

The Western Association of Technical Chemists and Metallurgists, which was organized in Denver, Col., six years ago, has taken the necessary steps to dissolve the incorporation and disband. The association was formed to promote the interests of Western men engaged in chemistry and metallurgy of the non-ferrous metals and has unquestionably supplied a need that was not met by any other society in the United States.

The conduct of the affairs of the association has never been an easy task, for the Western country is large, its centers of industry few and far between, and the men in position to give time and attention to the work are widely scattered. Further, concerted action often has been difficult by reason of the itinerant character of a large proportion of the membership.

The association has published six volumes of a creditable journal, which has been the most difficult feature to maintain, and without which it would have been impossible to have attracted or held a membership. It is with considerable regret that the board of control has taken steps toward disbandment, but the action was taken only after securing an expression of opinion from the members, whereby it was concluded unwise to enter on a new year with poor prospects of a successful finish.

The Utah Society of Engineers conducted an excursion over the lines of the interurban roads recently put into commission between Salt Lake City and Brigam City, Utah, on Saturday, Nov. 26. Stops were made to inspect power plants at Willard, Ogden Canyon and Lagoon; to see the concrete-lined Weber-Davis canal and to view other engineering works of interest. This is the second regular inspection trip made this season, the first having been to the new smelter of the International Smelting & Refining Company at Tooele, Utah, on Oct. 29.

Zinc Ore-Dressing in Colorado--IV.

By H. C. Parmelee.

Magnetic Separation.

The presence of much iron sulphide in zinciferous ores necessitates the adoption of different treatment than in the case of ores practically free from such mineral. Wet dressing alone is insufficient for the separation of the iron from the zinc on account of the slight difference in specific gravity, that of blende being about 4, and of pyrite 5. If galena also is present it may be separated by the usual wet methods, its specific gravity being 7.5.

In case all three of these minerals are present two courses are open. By preliminary wet dressing the lead may be removed as a clean concentrate and a zinc-iron middling made for further treatment by magnetic or electrostatic processes; or the treatment may be reversed, the iron being removed by magnetic separation and the zinc-lead product finally separated on tables. Electrostatic separation of iron and zinc is effected without preliminary roasting, and for the details and principles of this process the reader is referred to our issue of November, 1910 (Vol. VIII, No. 11, page 636).

Magnetic separation of blende and pyrite may be accomplished without preliminary roasting by the use of high intensity

WILSON MILL, ROBINSON, COL.

magnetic separators of the Wetherill type, the zinc mineral being lifted out of the mass. But the more usual procedure is to give the ore a magnetic or "flash" roast, rendering the iron magnetic, after which it is separated from the mass by separators of the low-intensity type, such as the Dings. Manifestly the choice of machines and method of procedure depends largely on local conditions and must be determined by experiment. In Colorado both types of machines and methods of practice are employed, although the prevalent method is roasting and magnetic separation, followed by wet dressing.

The object of the roast is to convert certain non-magnetic minerals into magnetic substances so that they may be separated by means of a magnet properly applied. The conditions required for successful work are very precise and need considerable care. Either magnetic sulphide, Fe_3S_4, or magnetic oxide, Fe_3O_4, may be produced, but some experiments have shown[1] that in an ore containing pyrite, blende and galena the magnetic concentrate runs lower in lead and zinc when magnetic sulphide is produced than when magnetic oxide is the

[1] G. M. Gouyard. *Proceedings Colorado Scientific Society.*

product of the roast. Furthermore, the production of magnetic sulphide is a simpler procedure than roasting to magnetic oxide, the latter usually requiring the addition of carbonaceous material to reduce ferric oxide to magnetic oxide.

Siderite often occurs in zinc-lead-iron ores, and in the calcination process it is converted into magnetic oxide under much the same conditions which convert pyrite into magnetic sulphide. Less time and slightly higher temperature are required for the conversion of the siderite than of pyrite, but an

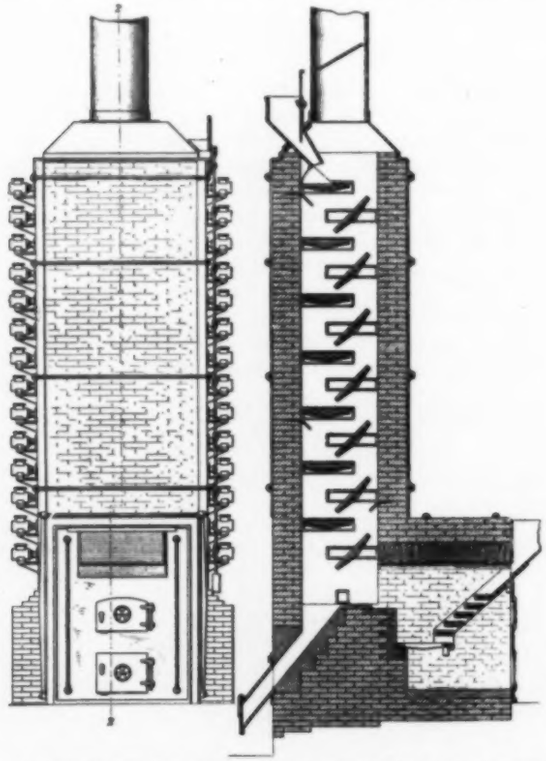


FIG. 1.—FAIRCHILD ORE DRYER, FRONT ELEVATION AND SECTION ON LINE 2-2 OF FRONT ELEVATION.

ore containing both may be treated successfully, although the same degree of conversion of both minerals can hardly be obtained. Ore properly roasted for magnetic separation will be recognized by its uniform black color and the absence of the reddish tinge which indicates oxidation. Blende will be coated with a thin layer of oxide. When these conditions are attained a magnet of low m.m.f. will remove the iron readily.

The Wilson Mining Company.

The mines and mill of the Wilson Mining Company are situated near Robinson, Summit County, Colorado. The ore is a mixed zinc-lead-iron sulphide, with some siderite, in a silicious gangue. The average composition is: SiO_2, 11 per cent; Fe, 23.6 per cent; Pb, 10.4 per cent; Zn, 14.8 per cent; CaO, 4 per cent; S, 35.6 per cent; Ag, 7 oz. per ton, and Au, 0.07 oz.

The mill has a daily capacity of 125 tons according to its present roaster equipment. The principal steps in the process are crushing and grinding, drying, roasting, magnetic separating and wet dressing. The mill is conveniently divided into units, each with ample bin room in which to store its finished product ready for treatment in the succeeding unit, thus making the operation of each unit independent of every other.

Crushing Unit.

There are no crude ore bins at the head of the mill, but ore is trammed from the mine directly to the crushing unit, where it is dumped over a grizzly (1½-in. openings) to a Symons No. 5C gyratory crusher, reducing to about 2½-in. size. This machine is much more compact than the usual type of gyratory, and differs further in having a stationary central shaft around which revolves an eccentric imparting the crushing force to the head. Automatic circulation of lubricating oil is provided by a pump, chain-driven from the pinion shaft. About 5 gal. of oil per minute are pumped to the top of the central shaft, where the stream is divided, half passing downward between the stationary shaft and rotating eccentric sleeve and half between the sleeve and crushing head. The oil collects in a basin in which the gear is housed and is returned by gravity to the pump.

The crushed ore and undersize of the grizzly are combined and discharged by belt conveyor over a revolving electromagnet contained in the pulley of the conveyor, for the purpose of removing pieces of iron and steel which accidentally find their way into the mill feed. The ore is then sized for a second crushing by passing over a shaking launder, 12 ft. x 3 ft., having a perforated area of 1-in. holes near its discharge end. The oversize is crushed to about 1-in. size in a Symons gyratory No. 3C.

A 14-in. B. and B. elevator then raises the crushed ore to a ¼-in. mesh trommel screen, the oversize of which is returned

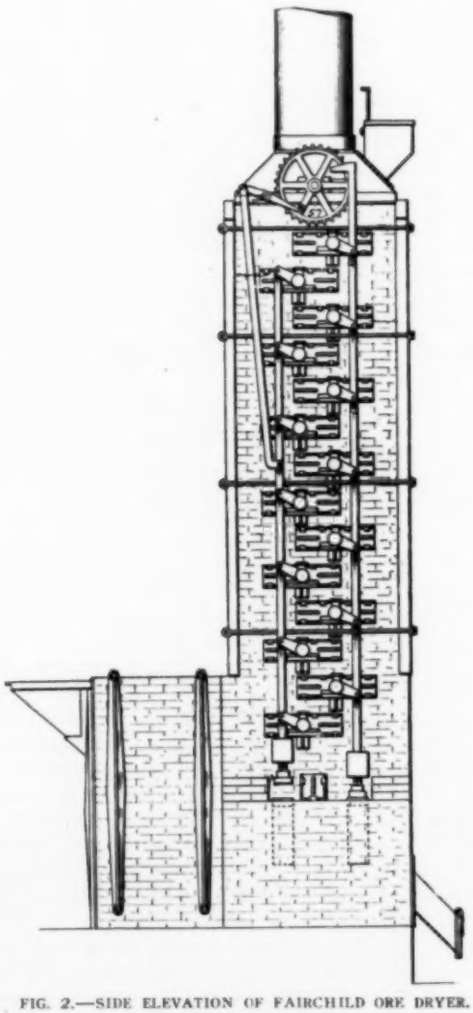


FIG. 2.—SIDE ELEVATION OF FAIRCHILD ORE DRYER.

to a set of 14 x 27 rolls set ⅜ in. apart. The roll product is returned to elevator and screen. The screen undersize, which is the finished product of this unit, flows to a point over the center of a rectangular ore bin of 1000 tons capacity, to which it is distributed by a belt running lengthwise of the bin. The belt and frame on which it moves is half the length of the bin and is operated by a reversible motor, so that ore falling onto the distributor may be dropped into the bin from

either end of the belt. The entire distributing mechanism also may be shifted by means of the same motor to occupy any position on a track extending the length of the bin. In this way ore may be conveniently distributed to any desired point.

Drying and Grinding Unit.

The crushed ore bin is provided with eight discharge gates through which ore is drawn for treatment in this unit. The gates are of the rack and pinion type, provided with a belt-driven plunger feeder. Ore thus discharged from any number of gates falls onto a belt conveyor running the length of the bin just below the gates. This manner of drawing the crushed ore from the bin insures a uniform feed for the succeeding unit, as regards size and contents.

The ore is elevated to the dryer, which is of novel construction, designed by Mr. O. H. Fairchild, superintendent of the mill. It is illustrated by drawings in Figs. 1, 2 and 3. It consists of a vertical brick chamber, about 20 ft. high and 5 ft. x 8 ft. in its other dimensions. Within the chamber are two sets of

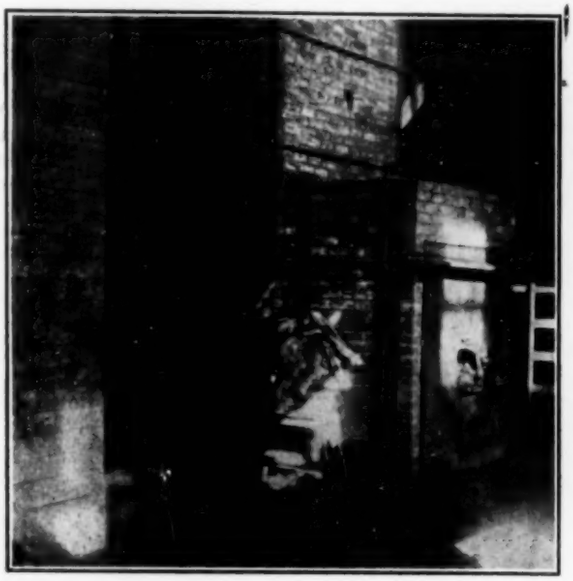


FIG. 3—ORE DRYER, WILSON MILL.

pans so arranged and operated that each set alternately dumps its contents onto the other. As the pans of one set are staggered vertically with reference to those of the other it will be seen that ore fed onto the uppermost pan makes its progress through the dryer by a series of intermittent discharges from pan to pan. This exposes new surfaces of ore to the hot gases ascending from the fire box in the base of the chamber and thus effects the drying in a short time. There are seven pans in each set. Ore is discharged from one set to the other every 15 seconds, so that it requires 3½ minutes for ore to pass through the dryer. Ore is delivered by elevator to the hopper shown in the top of the chamber. The feed is continuous, but the pans dump when a charge of about 75 lb. has been received, giving the dryer a capacity of about nine tons per hour. The fuel consumption is about 7 lb. to 8 lb. of coal per ton of ore dried. Cost of fuel, $4.75 per ton.

From the foot of the dryer the dry ore is conveyed by traveling plow conveyor through a metal launder to the elevator which feeds a 6-mesh trommel screen. The oversize is returned to a set of 14 x 27 rolls and thence again to the elevator and screen. The undersize passes to a second trommel fitted with 10-mesh screen, the oversize of which is returned to a second set of rolls similar to the first and thence to the elevator and 10-mesh screen. The 10-mesh undersize is the finished product of this unit and is sent by belt conveyor to the dry ore bins of 275 tons capacity.

Roasting Unit.

From the dry ore bins the ore is elevated to an oscillating plow conveyor placed above the roasters, which are arranged in two rows, one on either side of the conveyor. The ore is discharged into feed hoppers on each furnace, to which it is finally fed by plunger feeders.

There are five roasting furnaces of the circular multiple-

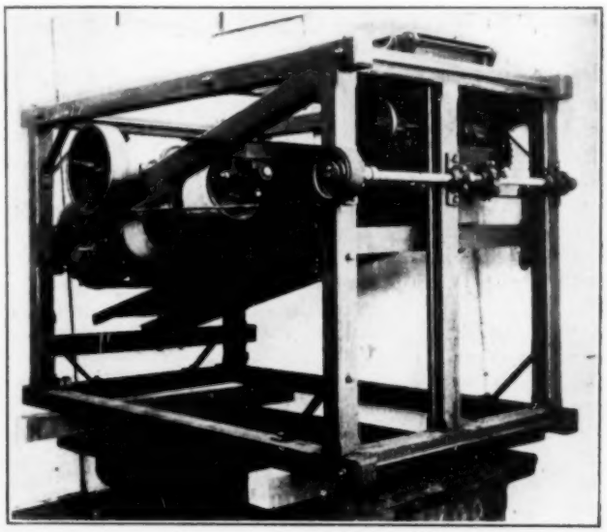


FIG. 4—FAIRCHILD MAGNETIC SEPARATOR.

hearth type, locally known as the Dewey roaster. Each furnace has six hearths, 14 ft. in diameter. A vertical central shaft driven from beneath the furnace carries the stirring arms, of which there are two, diametrically placed in each hearth. Ore is fed at the periphery of the upper hearth and plowed to the center, where it falls onto the next lower hearth and is plowed to the periphery, falling to the third, and so on through the furnace. The rabble arms make one revolution in 20 seconds, and the time required for the passage of ore through the

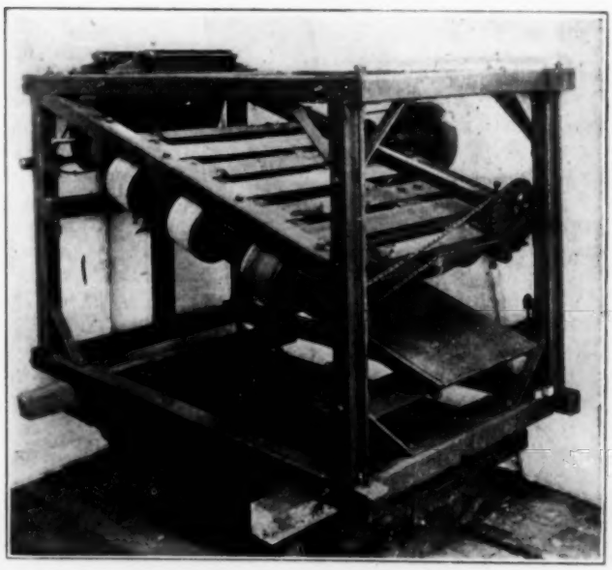


FIG. 5—REAR VIEW OF FAIRCHILD MAGNETIC SEPARATOR.

furnace is about two hours. The capacity of the furnace is 24 tons in 24 hours, making a total capacity of about 125 tons per day, which determines the present capacity of the mill.

The temperature of the three lower hearths of each furnace is indicated by Bristol electric pyrometers. An average tem-

perature of 1100° Fahr. to 1150° Fahr. is maintained, although the furnace attendant is not wholly guided by the pyrometer indication. The appearance of the ore on the lower hearth is a good guide to an experienced man, and this is often accepted in preference to the pyrometer reading and regulation made accordingly. The temperature is usually a good guide to successful roasting for magnetic separation, but changes in the physical condition or chemical composition of the ore are such that two furnaces apparently receiving the same feed may have to be run at different temperatures in order to get equally good results in the following magnetic treatment.

Furnaces of the type described are usually preferred for magnetic roasting on account of their low cost of installation, maintenance and operation, and the fact that they require less space than revolving cylinders. Considerable dust is made, and at the Wilson mill the roasters are provided with dust catchers. The roaster gas passes from the furnace through a down-take into a cylindrical iron chamber with conical top and bottom. The gas enters the settler practically tangent to its periphery so that a downward swirling motion is imparted to it. The dust particles settle into the conical bottom and the gas rises and is discharged into the air. As the roasting of the dust particles is practically completed they are returned to the lower hearth of the furnace and mixed with the discharging ore.

The roasted ore is cooled in a revolving drum, 39 ft. x 3 ft., set at a pitch of 1 in. to the foot, and making about 4 r.p.m. Cooling water is discharged over the drum from a launder placed above it. The cooled ore is elevated to an Edison 10-mesh screen for the purpose of removing the small quantity of oversize which results from the fritting together of particles in the furnaces. The oversize of the screen is reground in a set of small rolls and returned to the elevator and screen. The screened ore is then conveyed by a traveling plow conveyor to bins of 500 tons capacity.

Magnetic Separating and Wet Dressing Unit.

The magnetic separators, which also are designed by Mr. Fairchild, are placed just below the roasted ore bins, and receive a continuous feed through spouts leading to their feed hoppers. The machine is illustrated in Figs. 4 and 5. There are three sets of electromagnets, the second and third being of higher intensity than the one immediately preceding it. The voltage is 110, and the amperage 1.5, 3 and 6 for the first, second and third magnets respectively.

Each set is composed of 10 electromagnets of alternately reversed polarity, this being the principal feature of the machine. The magnets are arranged transversely to the shaking tray, and a belt travels around each set for the purpose of removing the magnetic concentrate from the machine. Ore is brought into the magnetic fields on an inclined shaking brass tray, on the under side of which are three bars of iron so placed as to be beneath the magnets. Passing under the first set of magnets the strongly magnetic particles of ore are attracted to the cross belt and removed from the machine. The same operation is performed by the second and third sets, where the weaker magnetic particles are attracted by the stronger magnets.

The object of alternating the polarity of adjacent magnets is to obtain a cleaner separation. In the process of magnetic separation it happens that more or less non-magnetic material, mineral and gangue is removed with the magnetic concentrate by being mechanically entangled with it. By alternating the polarity of the adjacent magnets in each set the particles attracted to the cross belt are caused to change position as they pass underneath the successive magnets, thereby offering the greatest opportunity for mechanically entrained matter to drop out of the mass.

At the Wilson mill there are five of these separators, each having capacity to handle the ore from one roaster. The shaking tray oscillates 325 times per minute. The cross belts have a speed of 350 ft. per minute. The power required for the mechanical operation of a machine is about ¼ hp.

The magnetic concentrate is removed by belt conveyors running beneath the floor of the separator room and finally delivered to storage bins. This product is useful as fluxing material for lead smelters, and valuable also for the small quantity of lead, silver and gold contained.

The zinc-lead-silica mixture is also conveyed from the magnetic machines by a system of belts to a central elevator which raises it to a point above the concentrating tables, where water is added and the pulp prepared for concentration. The pulp is mixed by a screw revolving in a launder, water being introduced at about the middle point of the latter. From the mixer the pulp flows into a box which provides for the immediate elimination of some slime by overflow. The heavy pulp is then classified in a pocket classifier, the pockets being conveniently arranged at points in a launder extending above the tables. Hydraulic water is introduced by means of a pipe entering the side of the pocket and standing vertically in it. The pipe is capped at the top and water is discharged through three annular series of holes drilled in the pipe. The first series is 1 in. below the cap and each of the others is arranged 3 in. below the one above it. Just above each set of holes an annular apron is attached to the pipe to spread the flow and provide a uniform rising current of water in the sorting column. The last two pockets are dewatering boxes with clear overflow and slime underflow, which is combined with the first slime for concentration.

The table equipment comprises nine Wilfley and two Card tables, one of the latter being a special slime table. Eight of the tables are used for direct treatment, and three for retreatment of middling. The two Card tables are used for the finest sizes, one on direct concentration and the other on retreatment of middling. The products are lead concentrate, zinc concentrate and tailing; the concentrates are laundered to filter bins of the usual type and the tailing to waste.

Products and Sampling.

The following analyses of concentrates represent average shipments:

Magnetic Iron: SiO_2, 6.5 per cent; Fe, 46 per cent; Zn, 3 per cent; Pb, 2.2 per cent; Au, 0.05 oz. per ton, and Ag, 4.5 oz.

Zinc: SiO_2, 7 per cent; Zn, 41 per cent to 42 per cent; Fe, 11 per cent; Pb, 6 per cent; Ag, 8 oz. per ton.

Lead: Pb, 67 per cent; Zn, 6 per cent; Ag, 34 oz. to 38 oz. per ton.

The tailing carries about 9 per cent zinc and 1.2 per cent lead and amounts to 15 per cent to 18 per cent of the ore treated.

Provision is made for automatic sampling of the crude ore and tailing. The former is sampled just above the crude ore bins after it has been screened through the ¼-in. mesh trommel. The sample represents 5 per cent of the ore. The tailing sample represents 1/230 of the pulp, and provision is made for collecting a separate sample for each eight-hour shift. The tailing sampler is a horizontal wheel with vanes on its periphery, against which the stream of pulp flows, causing the wheel to revolve. At one point on the periphery of the wheel is a ½-in. slot which receives a portion of the stream and conducts it to the sample boxes. The slot bear a ratio to the periphery of the wheel of 1:230.

Power, Water and Labor.

The mill is electrically operated throughout, power being received from the Central Colorado Power Company at 10,000 volts and transformed to 440 volts. The distribution of motors is as follows:

Crushing Unit: For crushers, rolls, elevator and screen, 75-hp motor; maximum demand, 43 hp. For distributing ore to bins, 2-hp motor.

Drying and Grinding Unit: For dryer, rolls, elevators, screens and conveyor, 75-hp motor; maximum demand, 32 hp.

Roasting Units: For roasters, cooler and elevator, 25-hp motor; maximum demand, 14.5 hp; on cooler only, 6 hp. For conveyor and elevator, 25-hp motor; maximum demand, 9.6 hp.

Concentrating Unit: Motor generator, 10 hp; maximum demand, 11.2 hp. For tables, conveyors, etc., 25-hp motor; maximum demand, 17.7 hp; or with centrifugal pump, 23.7 hp.

The water supply is from Ten Mile Creek, and is pumped by an electric driven duplex piston pump to a 50,000-gal. storage tank 180 ft. above the creek. This pump is operated by a 15-hp motor. The pumping station is also equipped with a centrifugal pump operated by a 50-hp motor. Some of the mill water is reused, viz., that from the concentrate bins. The bins are constructed so that the filtered water is laundered to a sump, from which it is returned to tanks in the mill by a centrifugal pump having a capacity of 180 gal. per minute and operated by the motor running the tables.

The labor required in the operation of the mill is five men per eight-hour shift when crushing is done, which is usually but one shift per day. On the other two shifts only four men are needed; one for the dryer and rolls, one for roasters and separators, one for tables and a shift boss.

The Wilson mill is the latest in Colorado to apply successfully the process of magnetic separation to complex sulphide ore. The foregoing analyses of the products attest the quality of the work, and it will not be long until the plant is working at full capacity. Acknowledgment is due Mr. O. H. Fairchild, general manager of the company, for the data and information.

Magnetic Specimen-Holder for Microscope.

A specimen-holder for microscopes, particularly a magnetic holder for metallic specimens used under a high-power instrument, has recently been patented by Prof. ALBERT SAUVEUR, of Harvard University. In order to properly examine a piece of metal it is necessary that the surface be held at a plane accurately perpendicular to the optical axis of the instrument. While it is possible to prepare a specimen with two parallel sides the operation is a difficult one, and this difficulty is overcome by Dr. Sauveur by the arrangement shown in the adjoining illustrations, reproduced from his patent (977,842, Dec. 6, 1910).

Fig. 1 is a plan view of the mechanical stage of a high-power microscope equipped with the magnetic holder and showing a specimen in place, Fig. 2 is a plan view of Dr. Sauveur's magnetic specimen slide with a specimen slightly removed therefrom, and Fig. 3 is a central sectional view showing the specimen in place beneath the slide.

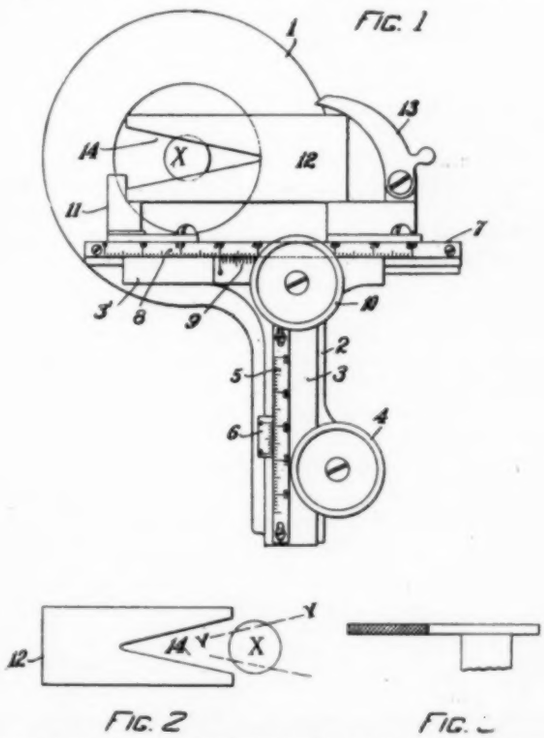
The circular stage 1 is provided with a lateral bracket 2 upon which is mounted a slide 3, being controlled by the milled wheel 4. The scale 5 reads on the vernier 6. The slide 3 carries a cross slide 7 having a scale 8 reading on a vernier 9 and controlled by a milled knob 10. On the cross slide 7 is a stop 11, having a right-angled notch, within which one edge of the magnetic specimen slide 12 may rest. The spring arm 13 bears against the corner of the slide diagonally opposite from that first mentioned, so that the slide is always clamped in exactly the same position.

The slide 12 is cut with a V-shaped opening 14, so that specimens of varying sizes may be held under it. As it is frequently necessary to re-examine a specimen at exactly the point first observed, Dr. Sauveur finds that this V-shaped opening affords the possibility for the following practice. A specimen, as indicated by X, is placed under the slide 12 in any convenient position where it has sufficient bearing to be held firmly by the magnetism of the slide and the operator then scratches on the plain surface of the specimen a pair of converging lines *Y—Y*, which will indicate the exact position on the V-notch 14 which the specimen is to occupy. The position will, therefore, be readily replaced at any time upon the slide and may be brought by the scale readings back to its proper position under the objective.

The Adhesion of Electrodeposited Silver in Relation to the Nature of the German Silver Basis Metal.*

BY ANDREW MCWILLIAM AND W. R. BARCLAY.

A few years ago the authors had their attention specially drawn to a rather serious trouble in connection with the behavior of high-quality electroplated spoons and forks under very severe conditions of wear. It is well known that at the present day a much larger proportion of such goods made are used in hotels, in steamboats and by caterers than formerly. Owing chiefly to the necessities of the case, but partly to the



FIGS. 1 TO 3.—MAGNETIC SPECIMEN-HOLDER FOR MICROSCOPE.

destructive instincts of virile humanity, these goods are subject to very much severer trials than they would have undergone had they found their way into use in the service of private families. For in hotels spoons and forks may have to be gathered up in large baskets and thrown into the receptacle in which they are to be washed, a kind of treatment that must be remembered in considering what follows.

A large quantity of spoons and forks, manufactured presumably under the best commercial conditions, examined and passed by experienced observers before being sent out from the factory, and apparently only differing from ordinary products in having an exceptionally heavy deposit of silver, were, after actual usage for eighteen months or so, returned to the makers with the complaint that the deposited silver coating was "peeling" or "stripping" along the edges and from parts of the surfaces of the articles.

As the trouble was not confined to a few isolated pieces, but extended to the majority of the articles in a consignment, and was also in time found to occur in connection with goods from different firms, it was felt that the matter warranted a thorough investigation, particularly as no specific fault could be traced in any of the processes through which these goods had passed, where the details of the processes could be known.

In order to correct a misapprehension which might arise, it may be necessary to point out that no attempt is here made

*A paper presented before the Institute of Metals at the January, 1911, meeting.

to discuss the ordinary causes of the "stripping" or "peeling" or "blistering" (as it is sometimes called) of electrodeposited silver from the basis metal during processes of manufacture, such, for instance, as that due to inefficient cleaning from grease or from oxide, careless treatment in scratch-brushing, or in the general preparatory stages of treatment, unsuitable plating liquid, etc., because articles so treated rarely pass from the hands of reputable manufacturers, the faults being revealed during the subsequent finishing and burnishing processes, it being well-known that if the slightest trace of grease or oxide remain on the surface of the basis metal the electrodeposited silver will blister under the pressure of the steel burnishing tools used in finishing.

The investigations were conducted from an entirely different point of view, the authors being convinced that the stripping of silver from articles which had passed all the ordinary tests of the manufacturer, and had indeed been in the hands of the users for periods varying from six months to two years, could hardly be ascribed to any of the ordinary causes mentioned.

From general experience and from the examination of many plated articles that had been a generation in wear the authors had formed the opinion that the adhesion had something to do with the nature of the basis metal, hence the investigations described in the present paper were undertaken with the view of determining not only the nature of the adhesion of electrodeposited silver to the alloys known commercially as German silver or nickel silver as a basis metal, but also to find out what difference, if any, exists between the various grades of German silver in general use in the trade, as to their suitability to form a basis metal for heavily electroplated articles which are liable to receive rough usage.

(A) Samples Tested.

The earlier experiments were made on a number of spoons and forks of various sizes supplied by manufacturers, and of the grades known in trade circles as firsts, seconds, thirds and fifths. Analyses of these were not made, but the makers reported them as having approximately the following compositions as to nickel: Firsts, 20 per cent; seconds, 16 per cent; thirds, 12 per cent; fifths, 7 per cent. The content of copper appeared to vary between 56 and 59 per cent, according to the few assays made.

In addition to these spoons and forks several test strips were obtained of each of the grades described, and of a similar character to spoon and fork blanks. All the pieces were treated so far as possible in exactly the same manner, and cleaned, prepared and electro-silverplated, according to the best methods in general use, with the one exception that a few of the samples were given a preliminary coating of copper before silverplating. The silver deposits given to these were equal to the highest qualities for high-class work, and the electroplating was done by experienced workers, under the ordinary conditions of works practice. The experiments on the above pieces were carried out several years ago, and samples were electroplated in different works. Those which will now be described have been made under the personal supervision of the authors during the last twelve months, as, in order further to test the matter, in view of the possible publication of results, they determined to prepare a series of nickel-silver alloys corresponding approximately to those in ordinary use among electroplate manufacturers.

The following table gives the compositions of the alloys accordingly made:

Table I.

Marks.	Nickel. Per Cent.	Copper. Per Cent.	Zinc. Per Cent.
A	20	57	23
B	18	57	25
C	16	57	27
D	14	57	29
E	12	57	31
F	10	57	33
G	8	57	35
BZ	18	50	32
BC	18	64	18

The choice of the above alloys is largely empirical, but they are fairly representative of the usual grades of copper-zinc-nickel alloys in commercial use. An endeavor has been made to systematize the series by fixing the proportion of copper on the uniform basis of 57 per cent throughout, with the exception of the last two members, BZ, in which 7 per cent of the copper is replaced by zinc, and BC, in which 7 per cent of zinc is replaced by copper.

Each alloy has been made under the ordinary conditions of works practice, manganese being used as a deoxidizing agent. The metal was cast into ingots of about 18 lbs. weight. These were rolled into sheets of a thickness and width suitable for spoon and fork blanks, and cut up into strips each 8 inches long and varying from ½ in. to 1 in. in width.

In every respect these strips correspond to spoons or forks with the exception that they have not undergone the process of stamping. A few odd strips from some members of the series have been made into forks, and these have been submitted to tests in the same way as the blank strips.

The electro-silverplating of these strips has been done under the best commercial conditions of manufacture, in an electroplating shop through which large quantities of spoons and forks are daily passing. They have been prepared for the depositing vat with due care and with all the precautions which would be taken by any careful electroplater to ensure the best adhesion of the silver deposit, but they were not subjected to exceptional treatment.

The electroplating solution used was the usual standard one of the double cyanide of silver and potassium, and the weight of silver deposited on each strip was at least equal to that given to the highest qualities of spoons and forks and to those previously described.

Table II gives the weight of deposit on the strips of each alloy, calculated per square inch of the surface.

Table II.

Marks	Deposit per Square Inch in Ounces Troy.
A	0.0196
B	0.016
C	0.0215
D	0.021
E	0.017
F	0.024
G	0.024
BZ	0.0198

The slight variation in the weight of deposit is explained by the fact that it was considered advisable to give each strip a minimum deposit at least of the quality indicated above, without in any way interrupting the course of deposition, such as by taking the specimens out of the vat for intermediate scratch-brushing, a practice often followed in the making of very heavy deposits in order to keep the coating as smooth as possible. The strips were consequently in the vat during the greater part of the working day, and naturally the rate of deposition varied as more or less work passed through the vat in the course of the day's operations.

(B) Tests Applied.

The guiding principle in designing tests for the purpose of the investigation was to subject the samples to strains such that the basis metal and its silver coating would be given, if possible, a certain amount of movement relative to one another, and in addition that the coating itself should be fractured. In all the cases of stripping of which mention has been made in the introduction the stages of development of the fault have appeared to be, first, that along the edges of the spoons or

forks concerned the silver deposit has in course of severe wear become broken or cut through, then these broken edges have gradually curled or peeled up, and if the deposited silver is thick, and therefore of sufficient substance, it may readily be pulled away from the basis metal surfaces. In some cases it has been possible by care and patience to peel the silver coating away from the fork almost from one end to the other.

With these points in view several preliminary tests were made with the apparatus available, and finally it was decided to adopt the test which had first appealed to the authors,

FIG. 1. PHOTOMICROGRAPH OF FIRST-GRADE GERMAN SILVER TEST-PIECE, COPPERED AND SILVER-PLATED. MAGN. 140 DIAM.

namely, a simple cold bending test. One part of the sample is gripped in a vise and the free part is bent to and fro until it breaks off. It is advisable that the angle of the bend should be about a right angle, in order to insure that the deposit and the basis metal get sufficient movement relative to one another.

It will be evident that this test is an extremely severe one, and that unless the metals are holding together almost as one the silver must be parted from its base, for the silver on the outside of the bend must move a greater distance than the basis metal, and on the inside of the return bend a less distance.

In view of the results it should be noted also that the order of increasing ductility of the metals being as a rule from the high-percentage nickel alloys to the low ones, the latter bearing more bending before fracture than the former, the test is more severe for metals of the G, F and E types than for A and B. In practically every case the samples were tested to destruction.

In addition to the mechanical tests applied representative sections were cut from each series for microscopical examination.

(C) Results of the Tests.

In the case of all the samples of the third and fifth grades and of the G, F and E specimens of the special alloys in Table I, it has been exceedingly difficult to peel up the silver coating in any way. In some few cases just at the point of fracture it has been possible to raise the silver coating very slightly at the top, but, in endeavoring to strip it further, the silver, though very tenacious and ductile, has broken rather than leave its base. It has, indeed, repeatedly surprised the authors to find how extremely adherent the coating is even under the severest tests.

Test pieces of first and second grades and of A and B of Table I, however, behave in a very different manner. In practically every instance the silver coating left the base at the point of fracture, and in a large majority of instances it could be pulled away quite readily from a considerable part of the surface without breaking the silver. In the cases of C and D evidence of imperfect adhesion was not so pronounced as in A and B, still the authors do not regard these as so good as E, F and G.

In the earlier experiments described a few samples of first and second grades were given rather a different treatment; they were prepared for plating in the ordinary manner, but before silver was deposited on them they were given a thin coating of copper from an alkaline copper solution, then transferred to the silver vat and given a heavy deposit of silver. These samples were then put through the same tests as the previous ones, and again there was obtained evidence of imperfect adhesion, though not quite to the same extent as formerly. The adhesion even in these cases was not to be compared, however, with that obtained with the lower-percentage nickel alloys G, F and E.

The electrodeposited silver in all the cases was tough and strong.

In order to carry the matter still further the authors quite recently obtained from another firm of German silver manufacturers a few spoons and forks of first and second grades, and the results given by these pieces when plated and tested are in every way confirmatory of the foregoing results.

A considerable number of test-pieces after plating and before being submitted to the bending tests were burnished by experienced silver finishers. This process is considered in the trade to be, in addition to a help in finishing, a reliable means of testing the adhesion of the silver. The operation is usually done by clamping the article firmly on a bench and then passing a smooth, dry steel burnishing tool over the entire surface and particularly on the edges. Unless the silver is in closest contact with its base at every point, the deposit will not pass this test without blistering. Every one of the test-pieces, spoons, forks and blank strips, used in the investigation emerged successfully from this test.

Small divergencies in results were noticed between test-pieces made from nickel silver that had been deoxidized and those made from alloy that had not been deoxidized, the former giving better results than the latter for the same grades of alloys, but the number of the latter used were so few that attention is only drawn tentatively to the result, as the authors intend to make further comparative tests in the future.

Results of Microscopical Examinations.—Samples of every type were carefully examined under the microscope at about 40 and 400 diameters, and in special cases at 1000 diameters.

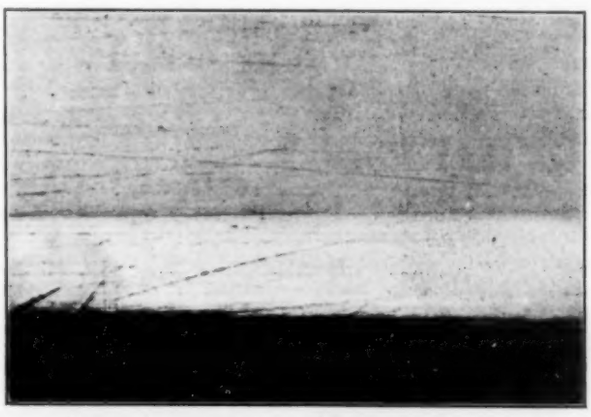


FIG. 2. PHOTOMICROGRAPH OF FIRST-GRADE GERMAN SILVER TEST-PIECE, SILVER PLATED. MAGN. 140 DIAM.

It was found that in every case, when not submitted to strain, the silver deposit appeared closely adherent, and the microscope did not reveal any marked difference between the nature of the junctions of the deposits with the various alloys. Those pieces from which the silver had lifted after being subjected to severe strain showed that the whole of the silver deposit had left the base metal, but in the case of the samples of firsts metal coppered before silverplating part of the copper had remained adhering to the silver and part to the alloy.

The photomicrograph shown in Fig. 1 represents a transverse

section of first-grade German silver coppered before silver-plating. The photograph includes a portion of the place where the mark had been struck, and shows the consequent movement of the two metals. The thin deposit of copper between the nickel silver and the silver is clearly discernible as a narrow dark band. This piece shows the three metals in closest contact throughout.

Fig. 2 shows a transverse section silver deposited on first-grade metal direct. In one place there is a slight indication of parting, evidently caused by the comparatively rough treatment by sawing and filing given during the preparation of the microsections.

Fig. 3 shows in transverse section of the third-grade metal dessert fork with a heavy deposit of silver. In this specimen, even at the highest powers, the two metals seemed to be in closest contact. The line of demarcation seen in the photograph is due to the difference in color between the metal and the alloy.

The scratches are more prominent in the photographs than the authors would have liked, but as the sections are quite sufficiently free for the purposes of their own examinations and records, they did not, when these were originally made, think of wasting any more time on them. When, however, it was decided that they should lay the matter before the members of this Institute, they repaired to the box where all the sections were stored together in order to prepare them more perfectly and a strange thing had happened.

The four most typical specimens which had been photographed had been left on the plasticene that had been used for temporary mounting purposes, and the sulphur that this useful material evidently contains had strongly attacked the samples during the three years or so they were laid away. Not only had the surfaces of all four sections been attacked, but the action extended along the junctions of the silver with the alloys so that the idea of re-preparing the samples until they were practically free from scratches was abandoned. This would really have been an unprofitable work, for the sections show all that is necessary. In the case of such a soft metal it is also a tedious one, for, in contradistinction to a recently published opinion, the authors find such materials as the comparatively hard tempered steels much more easy to polish than soft metals.

Conclusions.

The investigations clearly indicate considerable differences in the adhesion of deposited silver to the various grades of German silver alloys, even when all are treated under the same conditions of practice, and the authors are of opinion that this fact should have full consideration in choosing a basis metal for articles to receive the thickest silver deposits given, which will also be required to bear rough usage.

They are not prepared to state dogmatically that any one particular alloy is best for this purpose, but they consider they have fully proved that ordinary commercial alloys of copper, zinc and nickel containing more than 14 per cent of nickel are of doubtful utility in this respect. On the other hand, alloys of the types of G, F and E used for carrying heavy silver coatings electrodeposited with due care, will endure exceptionally rough usage without giving any cause of complaint on the ground of the silver stripping.

It should be noted that the tendency to strip increases with the thickness of the silver deposit.

The properties considered in deciding upon the grade of German silver to be used as a basis metal for electroplating are strength, color and malleability. As a general rule, passing from G to A in the series of alloys the strength increases, the color approaches more nearly the white of silver and the malleability decreases.

With regard to whiteness, however, where thick coatings of silver are concerned this should not be considered as a matter of such primary importance as the perfect adhesion of the silver under the conditions of wear. Strength is more important than color, and the lower members of the series are deficient in strength; but articles having quite sufficient strength and stiffness can be made of the type of alloy recommended by suitable mechanical and heat treatment.

With regard to the nature of the adhesion, the authors do not at present care to advance any theory to account for the differences described, but as they consider that the quicking process forms a superficial amalgam with the basis metal in the first place, and afterwards on the inner surface of the silver de-

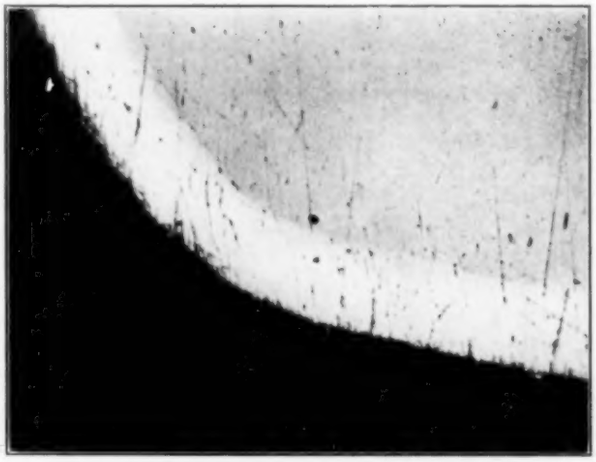


FIG. 3. PHOTOMICROGRAPH OF DESSERT FORK OF THIRD-GRADE GERMAN SILVER WITH THICK DEPOSIT. MAGN. 140 DIAMETERS.

posit, the two amalgams may form a kind of cement between the plating and the basis alloy.

It is possible that with the more suitable alloys this very thin film of amalgam is more efficient than with the alloys of higher nickel content.

Hence, although the investigations at present in train are undertaken with regard to the general properties and structures of the series of nickel silver alloys given in Table I, it is intended to note specially any changes in microstructure among the members of the series, with the view of any possible change in the nature of the amalgam that would be produced. They also, for the same reason, intend, with their new series, to examine, with the highest powers of the microscope, a fresh set of specimens after the most careful preparation. The above idea of the nature of the adhesion is supported by the manner of the tearing of the intermediate film of copper, part of which, as already stated, adhered to the silver and part to the basis metal.

A strongly worded appeal has reached one of us to support a demand for foundation alloys of higher nickel content to be used for the thickest coatings, which, if carried into effect, as in many other cases of undue interference of the buyer with the practice of the best makers, would defeat its own purpose of obtaining the best article for the required use.

It is interesting to note how the metallurgist has to reconsider the properties of his materials and his methods of manufacture in order to be able to make articles that will do him credit in the world's work when new conditions of life arise requiring these articles to bear a different type of usage. It is no use for him to claim that the usage to which his products are subject is unreasonable, if the new conditions of life require them to bear rougher treatment or some other special strain. He has no say in regard to sociological changes, and if he be wise he will just accept the conditions and endeavor to solve the problem. This was the task placed before the authors, and the practical results of their work have stood the severest trials since their earliest investigations were completed.

The Industrial Fixation of Atmospheric Nitrogen.

From a French Chemical Society Paper by E. LAMY.

Translated in abstract by MARK R. LAMB.

Before discussing the fixation of nitrogen it will be well to recall what nitrogen is, its importance and the rôle which it plays in nature.

Nitrogen is a part of all living matter, and plants as well as animals could not do without it while continuing to live and repair their tissues.

Nitrogen appears in nature in compounds, of which the most important is the nitrate of sodium, and in the free state in the atmosphere, of which it constitutes 80 per cent by volume, the balance of 20 per cent being oxygen. The atmosphere which surrounds the earth constitutes, then, an almost unlimited supply of nitrogen placed freely by nature at the disposition of plants and animals. It is true that the latter cannot directly assimilate the nitrogen of the air, but they borrow it from the plants upon which they feed. Now, though all plants require nitrogen, only some of them, such as the leguminous, thanks to certain bacilli, called "bactériocécidées" by Prof. Mattei, can directly fix atmospheric nitrogen. Very interesting experiments made during the last score of years have clearly demonstrated this.

Certain sea weeds have this same ability to fix nitrogen.

But the above classes constitute but a small part of the vegetable kingdom. The great majority of plants, such as wheat, barley, corn, oats and potatoes, upon which man and other animals are nourished, have not this faculty. According to recent experiments a small amount of assimilable nitrogen occurs in the atmosphere in the form of nitric acid and ammonia, but in infinitely small and altogether insufficient quantities for the needs of those plants.

In order therefore that they may grow it is necessary that they draw the required nitrogen from the ground. Now the nitrogen compounds which exist in the soil in its natural state are not sufficient to induce the best growth of plant life. So it is that it has become necessary to add nitrogen to the soil in the form of fertilizer such as manure and other natural products, of which the principal ones are the guano of Peru and the niter of Chile. As farming methods are improved and as farming becomes more intensive it becomes more and more necessary to enrich the land with nitrogen fertilizers to compensate for the elements removed from the soil by vegetable life.

Guano, which is an excellent fertilizer, exists only in small quantities, the deposits being about exhausted.

Fortunately this is not true of the niter deposits discovered in Chile about 1825. The first developments were made near the ports of Iquique and Patillos and afterward on the celebrated and inhospitable desert of Atacama. Now the plants are numerous and occupy an area of 500,000 acres.

The consumption of niter over the entire world is increasing at a rapid rate. The following figures will make this clear. The exportation of nitrates from South America was 800 tons in 1830; 7000 tons in 1835; 20,000 tons in 1850; 70,000 in 1860; 150,000 in 1870; 300,000 in 1880; 800,000 in 1890; 1,300,000 in 1900; 1,500,000 in 1905, and 2,000,000 tons in 1909.

At present the principal consumers of nitrate are as follows:

France	250,000	tons
Great Britain	100,000	"
Germany	500,000	"
Belgium	200,000	"
United States	400,000	"

The price of nitrate varies between 200 and 260 francs ($40 and $50) per metric ton (2204 lb.), and at present the price is 210 francs ($42) at Dunkirk.

The larger part of the nitrate goes to agriculture, but an important part is used in various chemical industries, among which the manufacture of HNO_3 should be given first place, since in France alone this industry absorbs ten to twelve thousand tons annually. Nitric acid is obtained from niter by decomposing the latter with heat by H_2SO_4. The uses for that acid are numerous and we may cite only the manufacture of explosives, of celluloid, artificial silk, dyes, etc.

The available reserves of Chile have been the object of many and widely varying estimates. Nevertheless the deposits are considerable in spite of the statements—perhaps intentionally pessimistic—which have been given out for some time.

According to a report given me by Mr. Paul Lemétayer, professor of applied chemistry at Santiago, a technical commission of the Chilean government has recently made a study of all the nitrate fields now being worked in order to determine the quantity of nitrate of soda which can be recovered by the processes now in use. After an exhaustive study of the fields the commission has arrived at the conclusion that even after prudently reducing the estimate the quantity of nitrate yet to be extracted amounts to 240 millions of tons. Therefore, even after taking into account the increase in the consumption, the supply is sufficient for agriculture and the industries for more than a century.

It should be added that, contrary to what is generally known, the deposits now being worked represent but a small part of the total area of the nitrate region, which covers about 25,000 square kilometers, or 10,000 square miles. This immense zone, which, not including unexplored areas, is over 700 kilometers, or over 420 miles long, will supply all demands for many years.

It seems possible to improve the present methods of working and particularly of treatment and separation of the nitrate from the foreign matter which accompanies it. The government of Chile has established an international competition, with important prizes for any one improving on the more or less primitive processes still in use.

Other deposits of nitrate have been found in California, Colombia and in the Caucasias, but up to the present none is important nor can the deposits be worked as easily as those of Chile.

There are, however, still other sources of nitrogen. For example, certain animal refuse, such as hides, wool, etc., spoiled fish and food, all of which are used as fertilizer after treatment with sulphuric acid.

Coal contains from 1 to 2 per cent of nitrogen, and as 800 to 900 millions of tons of coal are consumed annually in the world a considerable quantity of nitrogen is thus released. However, as at most but 1-20th of that coal is distilled in retorts of producers or coke ovens, and further as only a part of the nitrogen contained is recovered as ammonia, the amount of nitrogen compounds thus actually produced is small.

The recent development of new forms of coke ovens has appreciably increased the world's production of ammonium sulphate, so that this total may be given as 800,000 tons, of which more than 50,000 tons are produced in France. This is therefore an important source of nitrogen for agriculture, which will exist as long as the coal deposits of the globe endure, but it will not make up for the shortage when the natural nitrate deposits are exhausted.

Is, then, humanity, and the white races in particular, destined, after some centuries, to be unable to feed itself on account of the impoverishment of the soil and the failure of the crops? This is a melancholy view of the situation which the illustrious savant, Sir Wm. Crookes, took in 1899 and which he opposed with the statement: "Before we suffer a real dearth the chemist will have intervened and he will have put off the day of famine to a date so distant that we and our children and our great-grandchildren may live without worrying about the future."

These prophetic words of Crookes have been realized. The scientist has turned to the enormous supply of nitrogen contained in our atmosphere, and, thanks to electricity, that benevolent fairy which has again come to the assistance of humanity, they are beginning to exploit this "deposit" and to extract

nitrogen in a form which is useful to the industries and in agriculture.

We will now see how the problem of the fixation of nitrogen has been solved.

The nitrogen compounds obtained are in two forms:

First.—Cyanamide, or "lime nitrogen."

Second.—Nitric acid and the nitrates.

(*A*) *Cyanamide.*

This name is applied to a product composed of carbon, nitrogen and calcium, according to the formula CN_2Ca.

In organic chemistry "amide" is the name for a dehydrated salt of ammonia. Cyanamide CN_2H_2 results in this way from the dehydration of ammonium cyanate, $CNO.NH_4$, according to the formula

$$CNO.NH_4 = CN_2H_2 + H_2O.$$

Calcium cyanamide is a calcium salt of cyanamide CN_2H_2, in that the two atoms of hydrogen are replaced by one of calcium, as is shown in the above formula, CN_2Ca.

Franck, a scientist of Charlottenburg, discovered cyanamide by a lucky chance and without seeking it. With Caro he was studying the preparation of metallic cyanides, such as barium cyanide, when he discovered that the carbides of the alkaline earths absorb nitrogen under the influence of heat—that is, if nitrogen is passed through calcium carbide which is kept at a high temperature, the gas is absorbed; the carbide looses an atom of carbon and is changed to calcium cyanamide,

$$CaC_2 + 2N = CaCN_2 + C$$

In the presence of water (as, for instance, in moist soil) cyanamide breaks up into ammonia and calcium carbonate as follows:

$$CaCN_2 + 3H_2O = CaO.CO_2 + 2NH_3$$

The industrial manufacture of cyanamide was accomplished first in Italy at the Piano d'Orte plant near Pescara, where there are available 25,000 hp in hydroelectric power, then in Switzerland, in Norway[1] and in France (at Notre-Dame de Briançon).

The Société des Produits Azotés has installed an important plant near the carbide factory.

The carbide of calcium used is the ordinary commercial product, 80 per cent pure, yielding 300 liters of acetylene gas per kilogram (4.8 cubic feet per pound). It is crushed, then ground in a closed ball mill in which circulates a current of pure, dry nitrogen to prevent any alteration due to the humidity of the air. The fine product is charged into retorts similar to those used in German gas plants and is heated with coal, or, as is more frequently done, is charged into iron crucibles heated electrically, as at Notre-Dame de Briançon, the charge per crucible being 400 to 500 kg (900 to 1100 lb).

When the temperature is sufficiently high (cherry red), say, 1200° to 1300° C. (about 2200° to 2400° Fahr.), the current of nitrogen is introduced.[2] The absorption of the gas continues regularly and during 30 to 60 hours. The completion of the reaction is indicated by a manometer in the nitrogen pipe line.

As the combination which takes place is exothermic, the quantity of coal or of current required for heating the furnaces is relatively small.

The product obtained is a blackish, porous mass which is immediately crushed and powdered for delivery in 100-kg (200-lb.) sacks.

The percentage of nitrogen varies greatly, from 15 to 20 and even more. It depends on the quality of the carbide employed and also upon the purity of the nitrogen. It is important that the nitrogen should contain no oxygen and should be very dry. For these reasons the Linde and Claude apparatus is usually employed,[3] as this gives a product 99 per cent pure nitrogen.

As this apparatus has been described before let us now only consider the principle involved.

The air passes through towers sprinkled with a solution of caustic soda which fixes the carbonic acid. It is then subjected to a pressure of 4 to 6 kg per square centimeter (57 to 85 lb. per square inch), cooled with water at ordinary temperature and dried by circulating over calcium chloride.

It is then filtered to remove dust together with entrained particles of the reagents used.

The air thus purified is cooled with ammonia to at least 20° to 25° Cent., at which temperature it is put in the counter-current circuit which reduces it 180° to 190° in the coils surrounded by the gas coming from the distilling apparatus, and liquefaction at the valve results.

As in all processes of rectification (fractional distillation) it is necessary to obtain a very low temperature above the distilling apparatus in order to hold back the oxygen and separate the nitrogen, which is the more volatile.

This result is obtained by the use of a refrigerating machine, which consists of nitrogen pumps drawing gas from the upper part of the still, a circulating system parallel to the air circuit and the release of liquid nitrogen on the upper part of the still.

This nitrogen as it descends from one level to the next holds back the less volatile oxygen and thus pure nitrogen is obtained in the upper part of the apparatus with 75 to 80 per cent oxygen at the bottom.

The net cost of cyanamide depends first of all upon the cost of carbide since 80 lb. of the latter are required for every 100 lb. of cyanamide.

The use of cyanamide as a fertilizer in Europe has formerly disclosed several inconveniences. It could not be used as a soil covering. It had to be scattered at just the right time, either when the ground was being cultivated or when the seed was sown, as the very fine powder was blown about by the slightest breeze. It was also caustic and burned the hands of the farm laborers.

It seems now that a remedy for these disadvantages has been found in slightly moistening the cyanamide without allowing the temperature to rise, so as to prevent any heating which would transform part of the combined nitrogen into ammonia gas. This ammonia would not remain in the highly alkaline and hot mass and it would be released with what would represent a considerable loss.

After this moistening, which eliminates all traces of undecomposed carbide, the product is oiled with cheap hydrocarbons—petroleum in Germany, and coal oil in France—which gives a commercial fertilizer which is not dusty nor caustic.

A patent has recently been applied for mixing oxide of iron with cyanamide powder, which gives a granular product of good appearance and which it is easy to use.

(B) Nitric Acid and Nitrates from Air.[4]

We have now reached the second method, which is also the more interesting, that is, the production of nitric acid and nitrates from the nitrogen of the air.

Many processes have been designed during the last few years. We will not discuss all of these but will deal only with the three processes which are really practical and which are now in actual operation, and which consequently will interest engineers. These are the processes of Birkeland and Eyde, of the Badische Anilin und Soda Fabrik and of Pauling.

1. *The Birkeland and Eyde Process.*

In the apparatus[5] used by Birkeland and Eyde the principal and most important item is the furnace (Fig. 4 on page 74).

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[1] A large Norwegian calcium carbide and cyanamide works was described in some detail in our volume VII, pages 212, 309, 360 (May, July and August, 1909).—Editor.

[2] The temperature at which cyanamide begins to form is not more than 800° to 1000° C. (about 1500° to 1800° Fahr.). Since the reaction is endothermic, it evolves so much heat that no large supply of electrical energy from the outside is necessary.—Editor.

[3] In some cyanamide plants, instead of the liquid air process, a simple chemical method of burning out the oxygen from the air by means of copper is employed. The copper is mixed with asbestos to give a large surface and the oxide is reduced either with methane or with producer gas. The process is exceedingly simple on paper, but has given a good deal of trouble in practice on account of the excessive heat produced.—Editor.

[4] This part of Lamy's paper has been considerably abstracted to avoid repetition of what has been discussed at length formerly in this journal. Compare also the article on page 73 in this issue.—Editor.

[5] See also page 73 of this issue; further our vol. II, page 399; vol. IV, pages 126, 295 and 360; vol. VII, pages 304 and 305; vol. VIII, page 451.—Editor.

The arc is given a very great surface by means of the deviation produced by an electromagnet. The arc is thus caused to take the form of a flat disk with a diameter of 6 ft. in commercial furnaces. The general form of the furnace resembles a thick lens. It consists of two thick cast-iron shells which surround and support the refractory material. These are cored for air and gas circulation, and between them is a space 3 to 4 in. wide and 6 ft. long, which is the oxidizing chamber in which the electric discharge is maintained.

The electrodes consist simply of copper tubes bent in U-shape and cooled by a circulation of water so that their temperatures are never high. Fig. 4 on page 74 shows a vertical section of this furnace. The oxidizing chamber is represented by the restricted space between *a* and *a*. The electrodes, only indicated in the figure by the little circle in the center, are perpendicular to the plane of the drawing. The poles of the electromagnet are shown at the right and left, carrying the magnet coils. The current of air under pressure from a blower circulates in the furnace in the direction shown by the arrows. It enters through the two ducts at the right and left, shown at the bottom of the diagram, passing through two distributing chambers and through a large number of small tubes to the oxidizing chamber in the center. It is conducted from this through the annular canal *a a*, which surrounds the oxidizing chamber, finally escaping at about 1000° through the central exit duct to the apparatus which we will take up later.

It seems that the Birkeland furnace is remarkably durable, as it can run a year without excessive maintenance and repair charges. The refractories need replacing every five or six months and the electrodes every three or four weeks.

A number of furnaces of this type have been built at Notodden which vary in the amount of power required from 250 to 4000 hp.

The necessary electric current is supplied by the Svaelgos plant in the form of three-phase, 50-cycle, 10,000-volt currents.

2. Badische Company Process.

The engineers of this company, Dr. Schönherr in particular, in 1905 invented a different furnace system which is now being tested in the Norwegian plants, where it seems to be giving good results.*

The furnace of the Badische Company seems simpler than that of the Norwegian company. Its construction is cheaper, while all the costly and delicate mechanism necessary for creating the magnetic field is avoided, but it is more difficult to operate and requires more attention. The quantity of NO produced is practically the same, but the percentage of NO in the gas mixture is higher, that is, the effluent air contains 2 per cent of NO corresponding to 0.050 gram HNO_3 per liter (0.05 oz. per cubic foot), while the product from the Norwegian furnace contains from 0.03 to 0.04 gram per liter (0.03 to 0.04 oz. per cubic foot).

Several Badische Company furnaces have been built, but at the present writing they cannot use more than 700 to 800 kw, as already with this power the arc has reached a length of 7 m. (22 ft.). The two largest Birkeland and Eyde furnaces have a capacity of 4000 hp (2940 kw) each.

The average productions of the furnaces, which, as we have said, are practically equal, vary from 570 to 600 kg of HNO_3 (or from 12,500 lb. to 13,500 lb.) per kw-year. It does not seem possible to easily improve upon the latter figure. For a 750-kw furnace this represents 440 to 500 tons of HNO_3 per year, which is equivalent to 800 tons of commercial acid of 36° Baumé.

The Norwegian furnaces, which have been in use for several years at Notodden, have proved themselves eminently practicable. The Badische furnaces, which have been under test for some months, have behaved well up to the present, and it is difficult to choose between the two types, so much so that the Norwegian company intends to use both in the new works which it proposes to build.

But let us return to the gas which flows from the furnace of either system. This gas, consisting of a mixture of NO with a large excess of air, is, as we have said, at a temperature of 700° to 800° C. and even of 1000° to 1100° C. with the new 4000-hp furnaces. This temperature is much too high to permit of the gas being immediately conducted to the absorption towers. It is, therefore, cooled to 40° C. by being conducted through large boilers (the steam from which is used in evaporating pans which we will describe later), then through air-cooled aluminium pipes, the air thus heated being conducted to the furnace. Aluminium pipes are used because this metal is not attacked. The cooled gas is conducted to a sheet-iron tower lined with refractory material where the oxidation is continued and completed at the expense of the oxygen of the air. Then it is conducted to the high absorption towers, which are of two kinds. One is of Drammensfjord granite, not affected by acid, and with joints of asbestos and silicate of soda; the others are simply of wood.

The three granite towers are 34 m. (100 ft.) in diameter and 20 m. (65 ft.) high. They are filled with pieces of quartz which are continuously sprinkled with acid solution. The gas as it traverses successively the three towers is brought into contact, following a definite course, with more and more dilute acid.

In this way the strongest acid, which is taken from the bottom of the first tower, contains 500 grams of HNO_3 per liter, while pure water enters the top of the third tower. If the manufacture of calcium nitrate is the only object in view, it is useless to push the concentration too far, and in practice the acid is drawn off containing only 300 to 400 grams of nitric acid per liter.

The handling of this liquid is accomplished automatically by the use of montejus made of pottery which is not affected by the acid, and operated with compressed air.

At the gas outlet of the last tower about one-quarter of the NO is still undissolved. To catch this it is necessary to use an absorbent which is more energetic than water, such as a solution of soda or lime. This absorption is carried on in two towers filled with pebbles or pieces of granite. The absorption is so nearly complete that the gas discharged to the atmosphere contains but 4 per cent of the combined nitrogen, that is, 96 per cent of the NO has been recovered in the towers in the form of nitrogen compounds.

The liquid flowing from the wooden towers is a mixture of solutions of nitrite and nitrate of calcium if milk of lime has been used. This mixture is treated with nitric acid produced in the plant itself. The nitrous acid which is freed is conducted to the absorption towers.

If instead of milk of lime a solution of carbonate of soda is used as a spray in the wood towers, as is the general practice, the resulting solution is concentrated and the nitrate of soda is separated by crystallization and is a product in demand for the manufacture of dyes.

The nitric acid coming from the granite absorption tower is too weak to be put on the market. It can be concentrated to a commercial standard strength of 36° to 40° Baumé, and at present deliveries of this strength are being made in Norway. Eyde claims to have secured a 96 per cent concentration of nitric acid, but I have not yet been able to find that the Norwegian works have produced and sold acid so concentrated. They generally prefer to use the tower acid for the manufacture of calcium nitrate.

To do this, the acid is conducted to granite vats filled with coarsely broken limestone, which is rapidly attacked. When the solution is neutral—determined by adding a small amount of milk of lime—it is concentrated by evaporation in a Kestner apparatus, using the steam from the above-mentioned boilers, which are heated by the gas from the furnace. The material,

*This furnace is illustrated in Fig. 6 on page 74. For further details see also our vol. VII, page 245.—EDITOR.

kept in fused state to a temperature of 145° C. may be poured into thin-sheet iron cylinders where it solidifies upon cooling. It may be then sold in this way to chemical manufacturers, but it is generally solidified in cast-iron molds, crushed and pulverized and packed in 220-lb. barrels for delivery to the farmer.

The adjoining diagram is a flow-sheet of the above-described operations. 1 is the blower, 2 the electric furnace, 3 the boiler, 4 the cooler, 5 the oxidation tower, 6 the three absorption towers of granite, 7 and 8 the two absorption towers of wood, 9 the final nitric acid, 10 the nitrite solution. *A* is the apparatus for

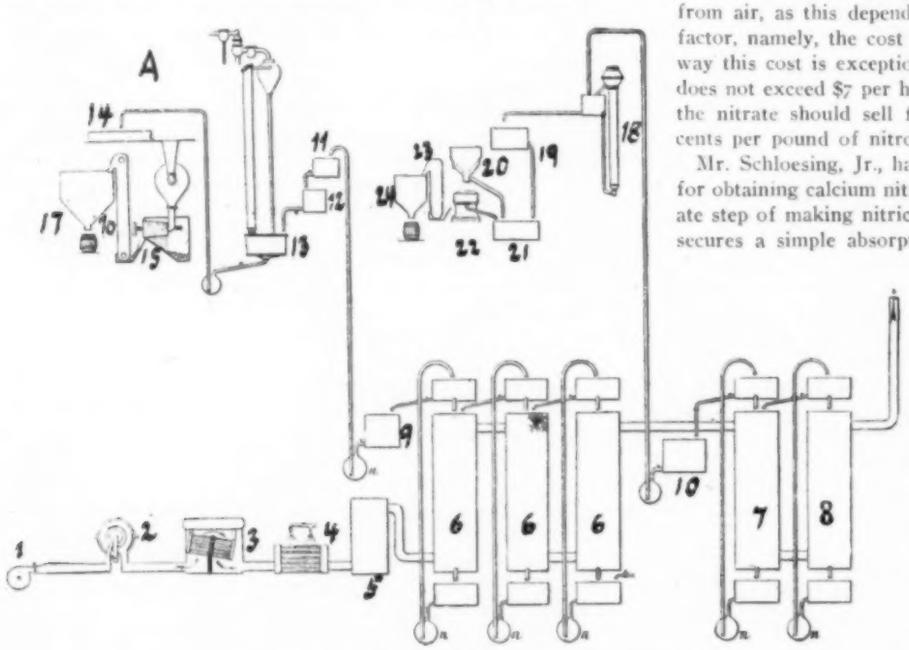


FIG. 1.—FLOW SHEET OF BIRKELAND-EYDE PLANT.

the production of saltpeter, *B* the apparatus for the production of nitrite, 11 solution tank, 12 a neutralization tank, 13 and 18 are Kestner evaporators, 14 solidifying tank, 15 crushing, 16 montejus, 17 receiver, 19 crystallization, 20 filter, 21 mother lye, 22 centrifuge, 23 montejus and 24 receiver.

The calcium nitrate thus obtained contains an average of 13 per cent of N and is an excellent fertilizer. The tests made both in France and abroad are conclusive in this respect.

Mr. Malpeaux, director of the Agricultural College of Berthonval, has begun a comparative study of sodium nitrate and the Norwegian calcium nitrate. During several years he has experimented with various plants, such as wheat, barley, sugar beet, potato, poppy and onion.

"We may conclude," Mr. Malpeaux remarked, "from our experiments that the calcium nitrate such as is furnished by the factory at Notodden at least equals sodium nitrate. While the former furnishes the plant with the calcium nitrate already formed, providing nitrogen and calcium at the same time, the sodium nitrate is changed in the soil to calcium nitrate and the remaining sodium salt is of no use to the plant. It is then in the form of calcium nitrate that the crop utilizes the nitrogen, and it is, therefore, not astonishing that under certain conditions the action of the Norwegian nitrate should be superior to that of sodium nitrate."

The opinions of scientists and farmers in other countries concerning the qualities of calcium nitrate seem to be the same as those of their French colleagues.

Using equal weights of calcium and sodium nitrates—from 100 to 300 lb. per acre, according to the crop—identical results were obtained, even though sodium nitrate contains 2 to 3 per cent more nitrogen, or 15 to 16 instead of 13 per cent nitrogen. In certain countries, especially in Germany, the prices of the two are the same. This is not true in France, where the nitrogen content is taken into consideration. If, for example, nitrate of soda is worth $1.90 per 100 lb. at Dunkerque, calcium nitrate sells for $1.70 per 100 lb. at Rouen.

The one slight disadvantage of the calcium nitrate is that it is hydroscopic, which makes it difficult to spread in the wet season, especially by the use of machinery. It is always best not to open the barrels until the moment when the material is to be used. Means are now being sought to diminish the hydroscopic nature of the product.

It is very difficult to state the net cost of producing nitrate from air, as this depends principally upon a very variable factor, namely, the cost of an electrical hp-year. In Norway this cost is exceptionally low, it being claimed that it does not exceed $7 per hp-year. Under such circumstances the nitrate should sell for $1.15 per 100 lb., this being 9 cents per pound of nitrogen.

Mr. Schloesing, Jr., has invented and patented a process for obtaining calcium nitrate directly without the intermediate step of making nitric acid as we have described it. He secures a simple absorption of the nitrogen oxide gas by lime. After having first dried the air entering the furnace, he conducts the gas leaving the furnace (cooled to 300° or 400°) to large sheet-iron chambers containing lime prepared in the following manner: Ordinary limestone is crushed and molded into briquettes and heated to the temperature of dehydration. These briquettes have the property of very easily absorbing the nitrogen oxide vapors at 300° to 400° C. He obtains a complete absorption of the NO and with an apparatus much smaller than is required by the other process. Moreover, the nitrate obtained contains 14.5 to 15 per cent of nitrogen. The process requires comparatively very simple apparatus and therefore entails a considerable economy in a given industrial application.

A small-scale test of the process has been made at Notodden which confirms all that Schloesing has claimed. There is therefore, reason to believe that a large-scale application will soon be attempted.

3. *Pauling Process.*

We will now describe our third practical process for making nitrates from air, that of Pauling.[1]

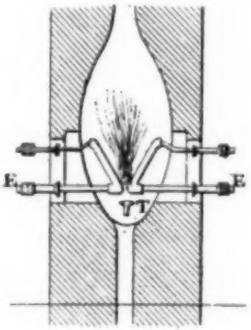
After numerous and long laboratory tests Pauling make his first installation in 1904 in the plant of the Saltpetersaure company, at Patsche, near Innsbruck, in Austria. Since then he has made a series of improvements on his process so that he now has made it a truly practical industrial process. In 1908 a French company, La Nitrogene, acquired the French rights to the Pauling process and is now exploiting it in a recently built plant at Roche-de-Rame, near Briançon.

We have visited this plant and we will now proceed to describe it from our own observations and from data kindly supplied us by the manager, Mr. Mollard, and the engineer, Mr. Gros.

Let us first go into the fine, large furnace room. The furnaces appear to be nearly square (3 ft. 3 in. by 3 ft. 8 in.), they are 10 ft. high and built in groups of three. Built as they are of simple masonry of refractory brick and stayed with iron, they show to a first glance two important qualities, simplicity and strength. The essential part is a vertical slot of refractory brick in which are placed the hollow cast-steel electrodes ar-

[1] See Fig. 3, page 73, of this issue; also our Vol. VII, page 430, where the Innsbruck plant is described.

ranged as shown in Fig. 2. They are reversible and interchangeable. Just below the space between these two electrodes and in the same vertical plane with them is the opening of a flattened tuyere *T* through which the hot compressed air reaches the furnace, the force of which serves to blow the arc. There is here a difficulty to overcome. It is necessary that the space between the two electrodes should be sufficient to pass the necessary quantity of air, but at the same time it should be constant and not extinguish the arc as it lengthens. Pauling has ingeniously supplied the main electrodes with auxiliary, movable copper knife-blade electrodes, the extremities of which can be brought near enough together to lure and start the arc as soon as it goes out, thus requiring a relatively low voltage.



The positions of the main electrodes are adjustable by means of screws, and of the knife-blades by a rack and pinion. These maneuvers are very easily made from without the furnace, the electrodes and their adjustments being protected by movable screens.

Under the influence of the jet of air from the tuyere the arc, rising between the copper blades, spreads to the tops of the horns of the electrodes and produces a flame 3 ft. to 4.5 ft. high. The combination of O and N in form of NO and nitrous vapors is made at the contact of the flame and the incoming air. In order to obtain a rapid cooling of the mixed gases, a suitable current of air is admitted at the bottom of the furnace. The furnace is provided with permanent peepholes for examining the knife-blades, electrodes and tuyeres, to assist in maintaining the furnace in certain and constant operation.

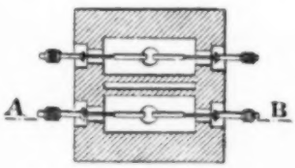


FIG. 2. PAULING FURNACE.

The copper blades last for 20 hours and can be replaced in two minutes. The electrodes are replaced with equal facility. They are good for 500 to 600 hours, thanks to the cooling effect of the circulation of cold water and also to the interesting peculiarity that the hottest part of the flame is the upper part, that is, the point at greatest distance from the electrodes.

An opening in the gas discharge channel permits the introduction of a pyrometer to measure the temperatures.

Each furnace includes two arcs in series using a three-phase 50-cycle 4000-volt current. The electrical equipment consists of two small coils, a small transformer and a starting box. All this is placed on the wall of the room in front of the furnace. The operation of the apparatus is simple and without danger. I have tried it myself. As soon as the switch is closed the furnace begins normal operation and one man can care for two batteries of three furnaces. Nine furnaces are now in operation. Each absorbs 500 hp, but it is expected to raise this figure to 600 and then to 700. The production will be proportionately improved, as was demonstrated in the Pratsche plant with furnaces consuming 1000 hp and even 1500 hp.

But, let us suppose that the furnaces are in operation. We have said that the air is brought to them through large conduits underneath, and that the air blows through the tuyeres, coming in contact with the long electric flame where the combination already explained takes place. The mixture of gases containing 1.15 per cent to 1.20 per cent of NO goes out at a temperature of 700° to 750° and immediately enters a cooling chamber, where it gives up part of its heat to the air which is on its way to the furnace. The gas is then conducted through two masonry flues alternately. While one is cooling the gas the other is connected to the atmosphere through a chimney, which cools it ready for use as soon as the other becomes too hot, which is usually after a period of six hours.

After being drawn through an exhauster which creates a suction as far as the furnace, the gas at a temperature of 35° C. enters a large oxidizing tower, made of reinforced concrete 35 ft. in diameter by 70 ft. high. It is in this tower, on account of its large volume, that a complete mixture is obtained and a prolonged contact provided and a complete oxidation of the lower nitrous vapors secured.

An earthenware conduit then carries the acid gases to the absorption towers, five in number. These towers are built of refractory brick and contain a material unaffected by the acid (similar to what is employed in the glover towers of sulphuric acid plants) over which flows pure water or water already acidulated in order to obtain a regular absorption.

A series of automatic earthenware montejus are used in handling these liquids.

The strong acid comes from the first tower. The acid reaches 5° Baumé in the fifth tower, 8° in the fourth, 12° in the third, 15° in the second and finally 25° to 30° in the first, which means 35 per cent to 40 per cent of nitric acid.

This strong acid is put in porcelain capsules, which are placed in a kind of Benker furnace, where it is concentrated by the hot gases which come directly from the furnace. The entrained water vapor is recovered from the cold gas in one of the absorption towers. The concentrated acid goes through a cooling coil of earthenware to large reservoirs, from which it is drawn for shipment.

This acid contains from 60 per cent to 62 per cent nitric acid and, therefore, measures from 38° to 40° Baumé, which is a good degree of commercial concentration.

The Innsbruck works, with special apparatus, is able, it seems, to push the concentration to 97 per cent nitric acid, which corresponds to more than 48° Baumé. It is also proposed to produce an acid of similar concentration at Roche-de-Rame.

Upon leaving the absorption towers the remaining gases are conducted through wooden towers containing soda solution and are then allowed to escape. A certain amount of nitrite of sodium is thus obtained, which, after concentration, crystallization and treatment in a centrifugal drier, is marketed in a crystal form, 97 per cent to 98 per cent pure.

Ninety-six per cent of the NO is recovered during normal operation. The yield amounts to from 65 grams to 70 grams (or 2.3 oz. to 2.5 oz.) of nitric acid per kw-hour consumed, or 1100 lb. to 1300 lb. per kw-year, or, again, about 2200 lb. of 36° commercial nitric acid, as in the Norwegian furnaces.

As regards net cost of production we must repeat that it depends largely upon the cost of power. The rather vague data on this subject which we possess does not permit our giving accurate information.

Taken as a whole, the Pauling furnace appears simple, compact, to require inexpensive repairs and the whole process should be practical; in fact, it appears that the Pauling plant particularly has reached a point where it runs regularly and with commercial success.

The Roche-de-Rame plant gives the visitor a curious impression, as do other similar plants. There are few men, no noise outside of the building of furnaces and, above all, no visible raw material, these being restricted to water and air, the entrances of which are not noticed.

But there is one very important element which enters quietly on copper and aluminium wires. That it plays an important part in the net cost can easily be seen in the above-quoted figures of production per kw-hour. It requires a kw-year to produce a single ton of calcium nitrate of 13 per cent nitrogen. Power must, therefore, be obtained in large quantities at a low price. Thus recourse must be had to "white coal" (the French name for waterfalls as producers of electrical energy).

At Roche-de-Rame water-power is obtained from the

Biaisse River, an affluent of the Durance. Four sets of 2000 hp have been erected and considerably more power is being sought.

But, as regards natural power, Norway is a privileged country. It is on this account that it has been considered feasible to create the new industry there, and that the powerful Norwegian Nitrogen Company, with a capital of $8,000,000, has built its plants in that country.

But Norway is not the only European country in which important available waterfalls can be found. In France, particularly, there are still good hydraulic powers available in the Alps and in the Pyrénées, and, even if the cost per horse-power of installation be appreciably higher than in Norway, it should be possible to reach a reasonably low net cost of production if the right to use processes can be obtained at a reasonable figure. It is, therefore, probable that other companies will follow the example of the Nitrogen Company and construct more plants in France to use one of the described fixation processes.

The fixation of nitrogen, due to the fruitful collaboration of scientist and engineer, constitutes one of the great discoveries of modern times—one of the most useful, undoubtedly, to agriculture.

That industry has no reason to fear the lack, even at a remote period, of the nitrogen fertilizers which it must have to support humanity. If even neither the niter deposits nor any of the other sources of nitrogen become exhausted, the farmer will find in calcium nitrate an inexhaustible supply of fertilizing material, because it will be possible to produce it so long as our earth is surrounded by its atmosphere and supply limestone—so long as water evaporates from the sea and, precipitating as snow and rain, forms torrents which descend from the mountains.

Thus will be realized a wish expressed by Reumaux, who said: "We should make the conquest of Nature by means of Nature herself."

Paris, France, and Lima, Peru.

Synopsis of Metallurgical and Chemical Literature

Iron and Steel.

Crystalline Structure of Iron at High Temperatures.—Walter Rosenhain describes in the *Physikalische Zeitschrift* of December 1, 1910, tests with iron and iron alloys to determine the nature of the change going on in iron and steel at high temperatures. As it was necessary to keep the original polished surface of test piece free from all oxides, even at temperatures above 1000° C., the iron was heated by an electric arc in a very high vacuum and tests were made in a specially constructed apparatus with a spring to test the effect of tension. Soft steels, electrolytic irons of great purity and almost chemically pure iron were tested. The conclusions may be summed up as follows: Iron up to 1100° C. acts like a crystalline aggregate and a plastic change of form takes place along the plane of cleavage of crystals of which it is composed. Between ordinary temperatures and 1000° C. iron exists in three modifications which have different mechanical properties. The temperatures within which these modifications take place can be identified with the three forms, alpha, beta, gamma, of iron according to the Osmond and Roberts-Austen theory of allotropic forms. The beta-iron forms at higher temperatures but is stronger and harder than the alpha-iron. It changes from the alpha-state into beta-state with an accompanying change in volume. Almost pure iron at high temperatures has the characteristic structure and a few characteristics of gamma-iron, as is also the case in several steel alloys. The existence of a hard beta-iron modification is also proven at high temperatures; failure of iron quenching is due to difficulty in preventing change from beta-state to alpha-state by this method. Evidence points to the fact that beta-iron if it could be kept unchanged at ordinary temperature would be very hard.

Direct Reduction of Iron Ores in the Electric Furnace.—Mr. G. Arnou in *La Revue de Metallurgie* December, 1910, discusses the new electric furnace of the La Neo-Metallurgic Company for the direct reduction of iron ores. The Chaplet-Neo-Metallurgic furnace is used, the design of which is shown in Fig. 1. It is an arc furnace; the current enters through one or more vertical electrodes and leaves through one or more side channels *B*. The latter are connected with the main bath *A* by horizontal channels *C* below the level of the bath. The slag is charged first, then the ore to be reduced mixed with carbon in proper proportions. Then the reduction begins under the action of the arc and the reduced metal filters through the slag and collects on the bottom. The furnace works similar to a small blast furnace where there is also a separate reducing and fusion zone. More than 15 tons of metal have so far been produced. The slag contains in normal operation less than 8 per cent iron oxide. The charge was a mixture of hematite ore and dried, powdered charcoal. Dried hematite briquets and wood charcoal and later mixtures of carbon and hematite in form of powdered dust were also tried.

With ore and carbon the furnace produced 77 lb. iron per hour.

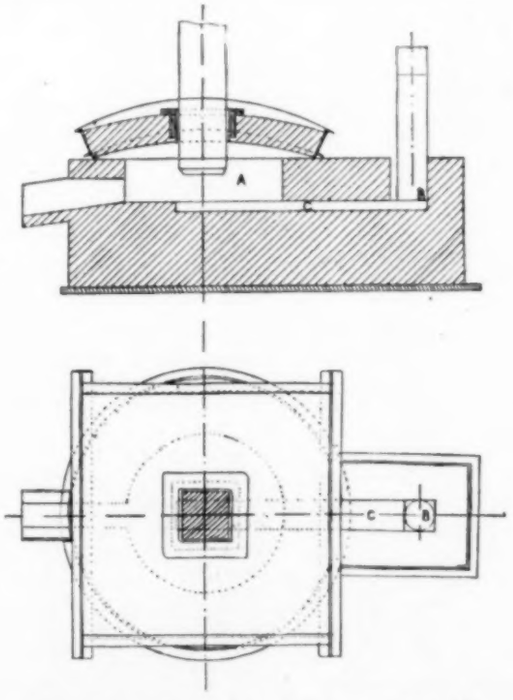


FIG. 1.—ELECTRIC FURNACE FOR STEEL FROM ORE

With briquets the furnace produced 70 lb. iron per hour.

With powder (dust) the furnace produced 68 lb. iron per hour.

The consumption of carbon was different in the three cases, being 660 lb. in the first and 726 lb. and 792 lb. per ton in the second and third cases. These differences are due to the hygroscopic nature of wood charcoal, the evaporation of the water consuming considerable heat. The differences are less if coke or anthracite is used instead of charcoal. In all cases it is possible to treat pulverized ores in this electric furnace, and even a mixture of hematite and retort coke in the powdered state without difficulty. Comparative tests were made, smelting hematite, magnetite, and siderite with coke, charcoal, and

anthracite. It does not make much difference which of the three forms of carbon is used. With charcoal the voltage tends to remain higher than with coke or anthracite, but this is not a serious defect. As regards thermal efficiency anthracite gives the best result. Reductions of hematite with charcoal produced 68 lb. per hour, while anthracite gave 84 lb. an hour with the same electric energy consumption. In the first case 792 lb. carbon per ton of iron, in the second case 596 lb. were used. Magnetite gives better returns than hematite with charcoal. Reduction of magnetite briquets produced 90 lb. per hour, whereas hematite gave only 70 lb. With anthracite both ores gave about the same result. The siderite was easily reduced, but with a large consumption of energy, yielding only 64 lb. per hour for the same energy consumption. Products obtained when analyzed gave the following results:

Carbon	Manganese	Silicon	Sulphur	Phosphorus
0.08%	0.10%	0.02%	0.02%	traces
0.09%	0.09%	0.04%	0.02%	"
0.11%	0.46%	0.06%	0.02%	"

In the last case a large addition of ferromanganese had been made for comparison. These results were obtained by direct reduction from the ores. The iron is perfectly malleable and can be hammered cold into thin sheets. After tempering it bends at 180° even with 0.20-0.25 per cent carbon, which fact is due to its small percentage of manganese. It has great tenacity. As to the cost, it is stated that most of these tests were made with a 120-kw furnace. In the reduction of hematite with charcoal for the production of soft iron the consumption was 3.430 kw-hours with a 120-kw furnace and 2.600 kw-hours with a 200-kw furnace. Studies in regard to the influence of the forms of ores and carbon compounds used as charge on the amount of energy used for the production of 1 ton of soft steel gave the following results:

Magnetite and charcoal.....3.150 kw-hours
Magnetite and anthracite.....3.050 kw-hours
Hematite and charcoal.....3.430 kw-hours
Hematite and anthracite.....3.100 kw-hours
Siderite and anthracite.....4.000 kw-hours

These tests were made in a 120-kw furnace. For a 200-kw furnace the efficiency is as high as 80 per cent.

As to the cost of maintenance it is pointed out that the arc in this furnace is entirely enclosed by the charge, which absorbs all the heat of radiation, so that the heat losses are small. The wear and tear are small. During the entire time of the tests the furnace used was not repaired and was as good at the end of operations as at the beginning. The amount of carbon in the charge varies with the different mixtures. Reduction of powdered hematite required 792 lb. charcoal and 594 lb. anthracite per ton of iron. Magnetite under same conditions required only 682 lb. charcoal and 572 lb. anthracite. As a rule 55-77 lb. of electrodes are consumed for the production of a ton of soft steel.

Gold and Silver.

Milling at Cobalt.—An interesting feature of ore dressing at Cobalt, Canada, is that the methods in use had to be devised especially for the ores of the district. No precedent could be referred to by the metallurgist and it is greatly to his credit that he has succeeded so well. In the *Mining Journal* (London) for Dec. 17 and 24, 1910, Dr. JOSEPH T. MANDY has given the results of his observations on the work at Cobalt. At present thirteen mills are in operation by electric power, and three more are in course of erection. Wet methods of dressing are adopted, and in some cases hydrometallurgy is practiced to the extent of the final recovery of the metals at the mill.

No two mills are exactly alike, but are the same in general principle. For illustration, the equipment and practice of the Coniagas mill may be taken as typical. The ore is crushed in Blake and gyratory crushers to 1½ in., then rolled to ¾ in. and screened in trommels. The coarse is concentrated in Harz jigs and the fine on Wilfley tables. The residue from these is crushed in 30 stamps, following which the pulp is thickened and treated on Deister and Wilfley tables, followed by a canvas slime plant.

Considerable difficulty was experienced in the early work in saving values from the tailings, owing to the escape of small scales of metallic silver and smaltite which seemed to have a great tendency to float away. By paying more attention to the slime recovery this loss has been largely averted. The average extraction in the district is probably 82 per cent. During 1909 the ratio of concentration was 39 to 1. The cost of milling was higher in 1909 than in 1910 owing to the use of cheaper electric power in the latter year. The cost of the work at the McKinley-Darragh mill was $1.98 in 1909, and is reported to be about $1.30 in 1910. Power formerly cost in the neighborhood of $150 per hp-year, and is now obtainable at about $50.

At three plants—the O'Brien, Buffalo and Nova Scotia—cyanide plants are operated in connection with the concentration. Dr. Mandy considers the success of this process as yet problematical. The Nova Scotia plant uses pan amalgamation and cyanidation, but no figures are available on the success of this system.

The concentrating plants all treat what is known as the low grade ore, containing 10 to 40 oz. silver.

The high grade ore, varying from 100 to 3000 oz. silver, is treated in smelters. The smelting ore is further divided into low and high grade; the first carrying from 100 to 600 oz. silver and the last from 800 to 3000 oz. The purchasers pay a sliding scale according to the silver content, and have various schedules for the arsenic content. United States smelters treated 2/3 of the tonnage in 1910, and Canadian plants treated about 32 per cent, the balance going to England and Germany.

The ores are silicious and make a good fluxing ore in silver-lead smelting. They also may be used for converter lining, but for this purpose there are several objections, viz., the arsenic fails to volatilize completely, and practically not at all after the first 10 min. of blowing, causing about 1.5 per cent of arsenic to appear in the copper; the slag is inclined to be thick; the charge effervesces owing to carbonates in the ore, causing high copper slags, sometimes as high as 15 per cent; alumina frequently has to be added to the ore for binding; and finally, the lime and iron in the ore tend to rob the lining of active silica, leaving less to do the work of slag production.

Milling Native Silver Ore in Mexico.—A combination of amalgamation and cyanidation for silver ore is described by WALTER M. BRODIE in the *Mexican Mining Journal* for January, 1911. The ores of the Batopilas silver mines are peculiar in that the clean base minerals, galena, blende and pyrite, seldom carry more than 4 oz. silver per ton. The values occur mostly as native silver, argentite and sometimes arsenical silver.

The ore is sorted into two classes, the high grade native silver ore which is treated without concentration, and the low grade milling ore which is concentrated.

The high grade ore is stamped through a fine screen. Some coarse native silver accumulates in the mortar and is removed and washed by hand prior to smelting and refining. The screened material is amalgamated in pans by the following process: The charge for a 4-ft. pan is 1000 lb. ore, 15 to 17 lb. mercury, 4.5 lb. cyanide of sodium, about 4 oz. lead acetate, 3.5 lb. lime, and 1 to 1 of stock solution. The solution in the pan then runs from 40 to 48 lb. cyanide per ton, and never below 33 lb. The ore is first ground for six hours without mercury, when it is added and grinding continued for 10 hours. The charge is then run into a settler where it is leached and agitated, mechanically and by air, for 12 hours. The settler is then washed four hours, and the amalgam and residues removed and washed by hand. The former is prop-

erly retorted, and the latter is sent to percolating tanks, referred to later.

The slime washings from the settler are thickened in a Dorr machine, the clear overflow of which goes to strong solution sumps, and the thickened underflow to Pachuca tanks. One of the latter is filling while the other is being agitated and drawn off. Blaisdell filters are used, four charges being put through in 8 hours. The tailings from the filter run about 9 oz. silver per ton.

The low grade ore is concentrated 50 into 1. The concentrate consists of pyrite, galena and blende, with native silver and argentite. It is reconcentrated into two classes, high and low grade. The high grade is amalgamated in a pan, adding mercury pound for pound of silver contained, and a slight excess. Grinding proceeds for 14 hours before amalgamation is started, this being continued for 2 to 5 days. Cyanide, lime and lead acetate also form a part of the charge, and the pan is decanted twice daily into the rich solution sump, the solution being made up again with stock solution holding 10 lb. KCN per ton. After 6 days the pan is discharged into a settler, where it is agitated mechanically and by air for six days, the solution being decanted once a day. The tailings from the settler go to the Dorr thickener, and thence to the Pachuca tanks.

The low grade concentrate, which forms the bulk of the product treated, is leached in the percolating tanks referred to above. The charge consists of from 35 to 60 tons of ore; the chemicals added being lead acetate, lime and cyanide. The latter is added in sufficient quantity to make 50 lb. KCN per ton. The solution is left on 24 hours, then drawn off and the tank allowed to aerate for 48 hours. Stock solution of 10 lb. KCN per ton is then run on and allowed to stand for 12 hours, after which it is withdrawn and the tank allowed to stand dry for 24 hours. This process is continued with 12 hour periods of percolation and aeration for two months, when 95 to 96 per cent of the value has been leached from the concentrate. The precipitated silver assays about 800 fine, and the refined product about 990 fine.

Classification of Tailing Pulp Prior to Cyaniding.—For the clean separation of sand and slime a number of schemes have been devised with greater or less degree of success in operation. In the October, 1910, issue of the *Journal* of the Chem. Met. & Min. Soc. of S. Africa, E. H. Johnson gives some results obtained with a plant of the type invented by W. A. Caldecott some time previous, and described by him in the same publication, August, 1909. The idea consists in classifying the pulp in primary diaphragm cone classifiers, washing the underflow in secondary diaphragm cones, finally delivering the separated sand to a horizontal revolving filter table where excess moisture is removed by vacuum, and the dried sand plowed off and transferred to cyanide tanks.

Mr. Johnson gives results obtained at the East Rand plant, where four 20-ft. sand filter tables are in use, with their complement of four 9-ft. primary cones and two 5 ft. washing cones. This plant has for some months handled the sand from approximately 4000 tons of tailing per day. A little over 50 per cent of the pulp is separated as a slime with less than 2 per cent remaining on a 200-mesh screen. The sand contains about 1 per cent slime as determined by elutriation, while 88 per cent is 200 mesh. Not only is the size separation efficient, but the bulk of the pyrite is retained in the sand.

The cubic space occupied by the sand and slime produced by this process is greater for the sand and less for the slime than under former conditions. This indicates that the interstitial spaces of the sand are more nearly true voids than under the old system which left more or less slime in the sand. Tests on 20,000 tons of sand indicate 23.5 cu. ft. to the ton. The extraction is accordingly more rapid, as shown graphically in the original paper. The essential difference between this system of diaphragm cone classification and all previous efforts in the same direction, is the maintenance of a large underflow of minimum moisture, the slime elimination being proportionate to the reduction of moisture in the underflow.

Gold from Clay Deposits.—*Metallurgie* of Dec. 8, 1910, brings a note on a new method of winning gold from clay deposits, due to J. J. W. H. van der Toorn (German patent 225,810). The processes now employed treat clay with water whereby a good deal of the gold is lost, as the separation of gold from clay is very difficult on account of the great cohesion of the clay. The new process tries to remove this cohesion. The gold-containing clay is spread over a large area, dried and kneaded. Then it is heated in a furnace similar to a brick kiln, till it assumes the hardness of stone. The rocklike masses are then broken in a stamp mill and the gold separated in the usual manner.

Copper.

Early Pyritic Smelting at Leadville.—An interesting article on the operation of the old Bi-Metallic Smelter at Leadville, is contributed to the December, 1910, *Bulletin* of the Amer. Inst. of Min. Eng., by Charles H. Doolittle and Royal P. Jarvis, respectively former superintendent and chief chemist at the plant.

Pyritic smelting of raw silver and copper bearing ores was commenced at Leadville by the Bi-Metallic company about 1892 in a remodeled lead smelter, known as the La Plata. After the success of the process was established a new plant was erected in Sept., 1893, and operated until the spring of 1900. Its success was based on the excellent facilities for obtaining suitable ores; heavy sulphides for the production of a preliminary low grade matte, and rich silicious ores for the second smelting and production of a high grade matte.

The plant consisted of three blast furnaces of different dimensions, but averaging about 36 by 163 in. at the tuyeres, and 6 ft. 6 in. from tuyere-center to feed floor. The jackets were supported on a bed plate, and the tuyeres were placed about 13 in. above the level of bottom brick lining, giving a crucible of the depth mentioned. The furnaces were provided with sumps lined with fire brick, attached to the middle of their long side. Matte and slag discharged continuously through the sump into settlers, from which slag flowed continuously and matte was tapped periodically. The superstructure of the furnace was built unusually high, and each had a down-comer to a flue in which most of the dust was settled.

A system of wet recovery was in use also. The hot dry gases were drawn from the flue and treated in a scrubbing tower, where an additional recovery was made. The flue dust proper comprised 3.5 to 7 per cent of the ore treated and consisted of Ag, 14 oz. per ton; Au, 0.12 oz. per ton; Cu, 2.3; Pb, 3.5, and Zn, 7.1 per cent. The slime from the scrubbing towers comprised 2 to 4 per cent of the ore smelted and consisted principally of lead, gold and silver, and the object of recovery was its sale to lead smelters. It carried Ag, 16 oz. per ton; Au, 0.11 oz. per ton; Cu, 2.4 and lead (dry), 6.3 per cent.

Sampling was entirely by hand, a tenth-shovel portion usually being reserved when unloading shipments. The portion cut out for the sample was reduced in quantity and size, and finally the usual buyer's, seller's and umpire samples prepared. The balance of the sample was reserved until settlement had been made.

Smelting practice comprised two steps, preliminary smelting of the raw ore into a low grade matte, and further smelting of this product with silicious ores to form a matte carrying about 300 oz. silver per ton. Formerly mattes had been made carrying several thousand oz. per ton, but better saving was made with a lower concentration ratio. The final product contained 14 to 23 per cent copper.

The raw ore-charge contained about 14 to 20 oz. silver per ton, 2 to 3 per cent copper, and an excess of iron over silica or insoluble matter of from 8 to 15 per cent. Thus a charge consisted of 2600 lb. ore, 225 lb. lime rock, 1400 lb. slag from the concentrating furnace, 200 to 300 lb. flue dust, and 325 lb.

coke. About 65 per cent of the sulphur was eliminated on the first smelting. The concentration, based on net ore smelted and matte produced, was only 2.44 per cent. The slag from this operation was clean to be thrown over the dump.

The matte produced from the green ore-charge was re-smelted with silicious ores. The ore from the Commodore mine at Creede gave the best results, although some Leadville silicious ores were used even if leady. Cripple Creek ores were unsuited and gave sticky slags. The fuel consumption, based on net ore smelted, was about 14 per cent.

The capacity of the furnaces smelting green ore varied from 125 to 175 tons of net ore per 24 hours. On a concentrating charge from 225 to 300 tons of charge were smelted in the same period.

Electrolysis of Iron-Containing Copper Solutions.—*Metallurgie* of December 8, 1910 contains a short note on the use of revolving cathodes in the electrolysis of impure copper solutions, according to a process of F. Lacroix (German patent 225,652). Impure solutions containing only 1 per cent copper can be treated. The electrolyte containing ferrous salts is stirred up by rapid rotation of the cathode and is driven at considerable speed through a perforated anode by centrifugal force. The ferrous salts are changed at the anode to ferric salts and passed outside. This prevents the ferric salts from reaching the cathode where they would otherwise react with the copper deposited there. The electrolyte enters the electrolytic tank at a point btween the revolving cylinder cathode and the surrounding perforated anode. The cathode makes about 40 revolutions per minute. The anode solution is rich in ferric salts and these are passed into coke towers or similar apparatus where they are treated with the sulphurous vapors from the roasting process. The ferric salts are again reduced to ferrous salts with formation of sulphuric acid. The solution of ferrous salts is then taken again to the electrolyzer.

Tin.

Concentration and Reduction Works.—The production of ingot tin, 99.8 per cent pure, is reported from Mount Franklin, near El Paso, Tex., by CHAS. A. DINSMORE, who contributes a description of the plant to the Dec. 31, 1910, issue of the *Mining Journal* (London).

Concentration is done by a combination of wet and dry methods. The ore is crushed and screened by the usual methods. The first concentration is by Harz jigs, the tailing from which is dried, reground, sized and treated on dry tables. Sizing for dry concentration is very important, and in this plant 10 sizes are made, from 20 mesh to 200 mesh, by means of the Keedy dry sizer. The work of the tables is reported to be excellent.

The tin concentrate is reduced in a reverberatory furnace, oil being used as fuel. The charge consists of one part pulverized anthracite coal and eight parts concentrate. Reduction requires from six to eight hours, and about one ton of concentrate can be smelted per day. Temperature attained, 2700° Fahr. Loss of tin from concentrate to ingot, about 0.5 per cent. The concentrate represents about 60 per cent of the original tin in the ore.

A Cornwall Plant.—In the *Journal* of the Chem. Met. & Min. Soc. of South Africa for October, 1910, C. F. THOMAS has published the details of tin concentration at a plant in South Crofty, Cornwall. The crude ore is hand sorted and finally crushed through jaw crushers and stamps. The latter weigh 1065 lb., drop 7¾ in., 102 times per minute. Discharge 1¾ in. to 2¼ in., through 20-mesh gunmetal screen. This screen costs twice as much as steel screen, but lasts 5 days as against 36 to 48 hours for steel. Stamp duty, 2.8 tons per 24 hours.

The pulp is classified by Luhrig classifier into four products; the first three classes are treated on tables and vanners, the fourth, or slime, being settled and then treated on vanners. The concentrates are calcined in Brunton furnaces, 15 ft. in diameter. Capacity, 4 tons to 5 tons of concentrate to a sweet roast in 24 hours. Fuel consumption, 1000 lb. to 1200 lb. coal. The arsenic recovered from the calcination is sold crude.

Wetherill magnetic separators are used to clean the concentrate. The first separator has six magnets, the first two being used to remove iron and the other four to remove wolfram. The iron product contains about 3 per cent to 8 per cent cassiterite, very little of which is free. The wolfram product averages 40 per cent to 55 per cent tungstic acid, and is treated with concentrated sulphuric acid for from five to seven days, after which the residue is dried and again magnetically treated with the recovery of from 10 per cent to 25 per cent cassiterite. The non-magnetic cassiterite is easily cleaned for market. The current used on the separators is 110 volts, with 1.5 amp to 3 amp for the weak magnets and 10 amp to 14 amp for the strong.

Zinc.

Separation of Zinc from Vapors Containing Zinc Oxide.—According to *Metallurgie* of December 8 Paul Schmidt and Desgraz (German patent 225,956) proceed as follows: The vapor containing zinc oxide is passed through scrubbers of the usual type where the gases come in contact with the purifying liquid. The combination of water and soap solution did not give satisfactory results since the sulphurous acid generally contained in the vapors is absorbed by the water, soluble zinc salts being formed, and the zinc oxide dissolved. This new process therefore uses oil (crude petroleum, tar oil, etc.), as any zinc sulphate in the gases is insoluble in these solutions. The zinc oxide is separated and afterward freed from the oil by means of presses.

Aluminium.

Alloys and Impurities.—The *Giesserei Zeitung* of January 1, 1911, brings an article on aluminium alloys and impurities. The natural impurities of aluminium are silicon, iron, titanium, and the alkaline metals, sodium, potassium. Calcium and titanium are found very rarely and are easily extracted. Titanium has a good influence on aluminium favorably if it does not exceed 3 per cent. An aluminium alloy with 2 per cent titanium offers tremendous resistance to tensile stresses. Iron and silicon give luster to aluminium and make it hard. Sodium and all other alkaline metals are bad impurities in alloys poor in copper. They offer no resistance to the destructive influences of dampness and air. Aluminium bronze has from 7½ per cent to 10 per cent copper. Above 10 per cent alloys become brittle. Aluminium with less than 50 per cent zinc gives a non-crystallizing mixture of metals which is so homogeneous that the two metals cannot be recognized under the microscope. Nickel increases its elasticity. Magnesium-aluminium alloys do not resist the destructive influences as well as other alloys. They are expensive but very light. An alloy of this type contains Al 95.5 per cent; Mg. 1.50 per cent; Cu. 1.50 per cent; Ni, 1.50 per cent. Manganese and chromium increase hardness so that with 50 per cent chrome the aluminium cannot be worked. Tin in small quantities makes it fragile. The alloys of aluminium flow more easily than aluminium and it was tried to use them in soldering aluminium. But the tests were failures due to the great heat conductivity and oxidation of aluminium. Tungsten hardens aluminium and increases its resistance to any destructive influences.

Grinding.

Chilian Mills.—In Russia the Chilian mill is almost universally used to crush ores preliminary to amalgamation. Its use represents a slow development from the practice of crushing ores by spreading them on the main thoroughfares for carts and other vehicles to pass over. In *Bulletin* No. 75 of the Inst. of Min. and Met., London, Mr. H. C. BAYLDON traces the development of crushing in Russia and gives the results of present practice. The following tabulation gives a summary of the type of mills used in five typical plants.

	Plant No. 1	Plant No. 2	Plant No. 3	Plant No. 4	Plant No. 5
No. of Chilian mills in plant.	6	4	6	4	6
No. of runners per mill....	2	2	2	2	2
Diam. of pan across bottom.	7'6"	8'9"	8'2"	10'6"	9'4"
Distance between centers of runners	6'	6'9"	6'6"	8'3"	7'6"
Width of face of runner, in.	10	10	10.5	10	12
Diameter of runner.....	5'6"	5'6"	6'6"	5'6"	6'6"
Weight of runner, lb.....	5400	5400	5400	5400	108,000
Revolutions of mill per min.	12	14	13.5	12	11
Mean travel of runner per min.	226'	297'	275'	311'	260'
Height of discharge, in.....	10	10	9.75	9.5	8
Screen Used.	1 mm slot.	1 mm slot.	¼ to ⅛ in. punched.	1 mm slot.	1 mm slot.
Water to ore ratio, by weight.	10:1	6.5:1	7:1	13:1	..
Duty per 24 hours, tons....	16	26	20	19	24
Amalgam collected—					
from mills.....	86%	81%	83%	85%	..
from plates.....	13%	18%	16%	15%	..
from sundry sources....	1%	1%	1%	....	..

No accurate data on power is available, but it is estimated at from 7 hp to 12 hp per mill. The following sizing tests show the work of the mills.

1 mm screen	Plant No. 1 %	Plant No. 2 %	Plant No. 3 %	Plant No. 4 %
+30	0.2	0.4	0.2	0.1
+60	2.41	14.84	3.2	1.3
+100	14.72	22.06	12.80	11.0
−100	82.67	62.70	84.80	87.6

As the main object in view is to crush the ore very fine and to obtain the highest possible percentage of gold by amalgamation in the pan itself the results are excellent. The low capacity of the mill is due to the purpose for which it is used, and little change will be made as long as this factor remains the guiding principle in the milling practice of the country.

A pair of tires will serve for the crushing of about 9000 tons of ore and will weigh about 7800 lb. per pair when new and 1000 lb. per pair when discarded. Dies weigh 5000 lb. per set when new and about 2200 lb. when discarded.

Inside amalgamation is standard practice. Clean-up is made once a day, and 6 lb. to 8 lb. of mercury are added to the mill during the 24 hours. Outside hard plates are used and kept under locked covers. They are cleaned once a month.

Hand feeding is usually practiced, but the author describes a new mill for which he has provided an automatic belt feeder. The belt receives ore from a chute and is moved toward the mill a certain distance with each revolution of the runners. The device is automatic and increases or decreases the quantity of ore as needed.

The new Chilian mill, also devised by the author, which is used in connection with the automatic feeder, has runners 6 ft. in diameter, 1-ft. face and weighing 10,000 lb. It makes 14 r.p.m. to 16 r.p.m., the runners having a rate of travel of 340 ft. and 427 ft. In the mills running at 14 r.p.m. it has been found that a height of discharge of 9 in. to 10 in. and a water ratio of 4.5:1 to 5.5:1, with 1-mm screens, gives the best product and results. The wear of tires is from 0.62 lb. to 0.65 lb. per ton crushed, and of dies, 0.28 lb. The life of the slotted screen is about 1200 tons of ore crushed. The capacity of the machine is 42 tons per 24 hours.

The author believes that if the same amount of thought and attention were devoted to the development of the capacity of this type of Chilian mill as has been devoted to the heavy stamp-tube mill combination in South Africa it would prove a serious rival and give a product nearer to the ideal aimed at on that goldfield.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Blast Furnace Charging Apparatus.—With the object of regulating and evenly distributing the charge, and avoiding the abrasion of the inner wall of the furnace by the falling charge, Mr. J. B. McKennan, of Pueblo, Col., has devised the charging apparatus shown in the illustration, Fig. 1. A metal stock guide 1 with stock bell 3 and apron 3 form annular walls which guide the descending charge, causing it to fall in rings and preventing it from striking the wall of the furnace. The main stock hopper is shown at 5 and the receiving hopper at 8, the latter being closed by the bell 9. The distributing hopper 10 is provided with a spout 12, eccentric to its axis, and is designed to rotate above the receiving hopper. This movement

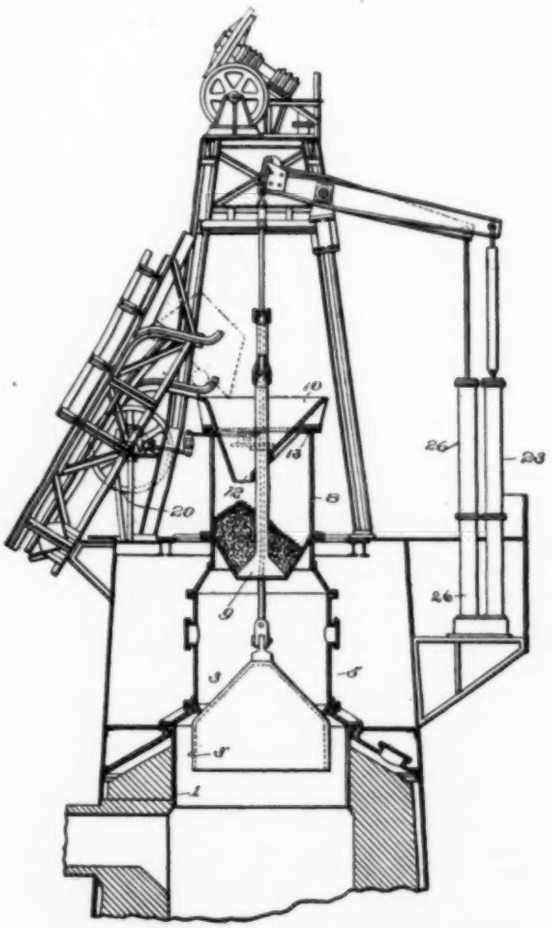


FIG. 1.—BLAST FURNACE CHARGING APPARATUS

is accomplished by means of rack and pawl mechanism connected with one of the sheaves 20 of the cable hoists.

In operation each skip load of material is delivered through the distributing funnel to the receiving hopper, as shown in the cut, the bulk of the charge being deposited eccentric to the axis of the hopper. The bell 9 is then lowered by the power cylinder 23, and the charge is deposited on the bell of the main hopper. The bell 9 is then closed again. During this interval another skip load is being hoisted and the funnel 10 has been shifted to another position, so that the eccentricity of the next load is different in position. Each succeeding load is advanced in position in a similar manner. In practice Mr. McKennan proposes to deposit nine skip loads of material in each furnace charge, six of coke, two of ore and one of limestone, preferably in the order mentioned. The distributing funnel will have seven positions and be moved two-seventh of a

revolution after each skip load has been deposited. Thus in seven furnace charges each skip load of the charge will have been put through the funnel while in each of its seven positions. After a full charge has been deposited in the hopper 5, the bell 3 is lowered by the power cylinder 26, and the charge falls through the annular opening formed by the stock guide and apron of the bell. (980,811, Jan. 3, 1911.)

Gold and Silver.

Agitator.—In the agitation of ore pulp it is desirable to have air valves which permit of flow in one direction only, so that should the air pressure be cut off for any reason, pulp cannot return into the nozzle. Such a nozzle is described in specifications of a patent granted to LOUIS T. SICKA, of Milwaukee, and assigned to the Allis-Chalmers Co. The details of the nozzles and their application to the bottom of an agitation tank, are shown in Fig. 2. There may be as many nozzles as desired, arranged around the bottom of a tank; one set projecting through the tank wall into the central well, and the other flush with the wall. In operation the set projecting into the central well cause the pulp to flow upward through it, and cause the agitation. The other nozzles are useful only in breaking up accumulations of slime or pulp when starting the agitator.

The pipe forming the nozzle 10 is capped at the end and perforated with a series of openings 11, which are larger at

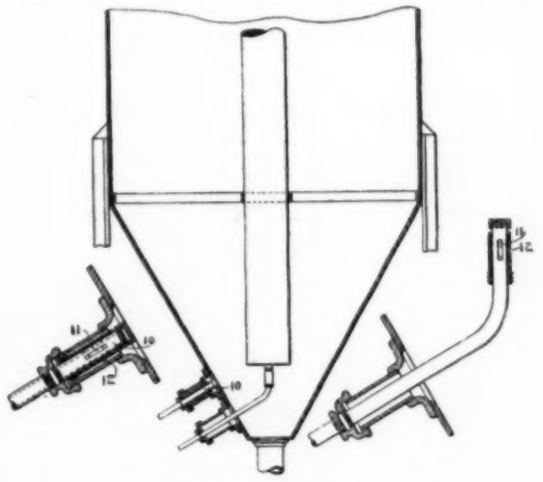


FIG. 2.—AGITATOR

the outer surface of the pipe. A piece of rubber tubing 12, is placed on the pipe over the openings, and fastened to the pipe at its lower end by cords. When air is forced through the pipes, it will press the rubber tubing outward and escape into the tank. If the pressure of air is cut off, the material above and outside of the nozzles will close the same and prevent the entrance of any foreign material. (978,667, Jan. 3, 1911.)

Suction Filter.—A continuous filter for cyanide pulp, which is also automatic in its action has been devised by HARRY E. KIER, of Colorado Springs, Col. The device is a circular rotary structure, upon which are mounted horizontal filter trays to receive and filter the pulp. In one revolution of the structure each tray is in turn brought into connection with suction for removing the cyanide solution, then wash water and suction to remove the weak wash solution, and finally with compressed air to loosen the cake and assist in discharging it when the tray is automatically dumped.

The filter is shown in vertical and horizontal sections in Figs. 3 and 4 respectively. As shown, there are eight filter trays 24, each connected by flexible hose with a set of pipes 51, 52 and 53, these being connected by means of valves 54, 55 and 56, with the circle tubes shown in the center of Fig. 4. The circle tubes have connections through the central hollow column with suitable pumps for creating vacuum or pressure. The pipe 51 is the suction pipe and is connected with suction pump 95; through it the first cyanide solution is withdrawn. The pipe 53 is a second suction pipe, connected with a suction pump 94, by means of which the weak wash solution is drawn off. The pipe 52 is the air pipe which delivers compressed air to the filter tray from a compressor, such as 96.

In operation the trays pass under a source of flowing pulp, as the pipe 47. After receiving a charge of pulp the valve

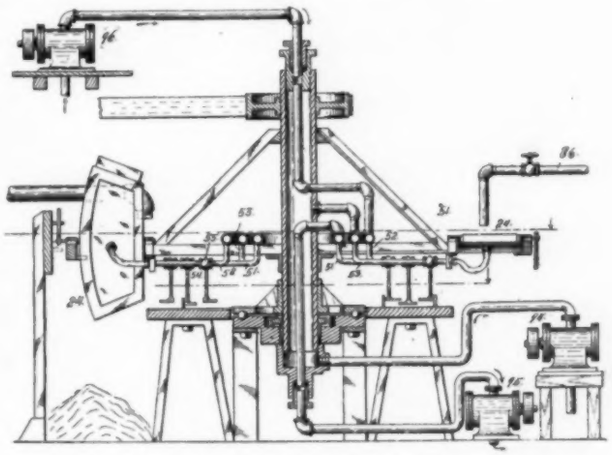


FIG. 3.—VERTICAL SECTION OF SUCTION FILTER.

connecting with the first suction pump is opened and cyanide solution drawn off. This suction continues until the tray reaches the wash water pipe 86, when it is cut off, and the cake subjected to a wash of water to remove the remaining traces of cyanide solution. From this point the valve connecting with the second pump is opened and the wash solution drawn off into a suitable receiver. Finally, the tray reaches

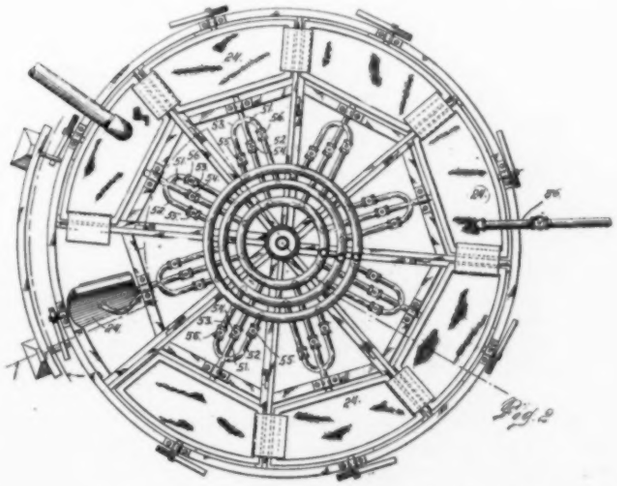


FIG. 4.—HORIZONTAL SECTION OF SUCTION FILTER.

a point where the suction is cut off and compressed air delivered into the filter tray under the cake, thus loosening it ready for discharge when the tray is dumped as shown in the illustrations. The action is automatic, the various valves being opened and closed by means of triggers mounted on the stationary platform 7, and engaging the stems of the valves as shown in Fig. 3. (978,381, Dec. 13, 1910.)

Copper.

Reverberatory Furnace.—In a patent recently granted to Messrs. W. J. MURPHY and A. E. CULLEY, of Butte, Mont., a new form of reverberatory furnace is suggested, the objects

being to provide a different method of feeding the charge, and a more efficient method of applying the heat. As shown in Fig. 5, which is a plan view, it is proposed to feed the charge at two constricted portions of the hearth area, through a system of hoppers and screw feeders, delivering the material at a point slightly above the level of the molten mass on the hearth. The inventors believe that this is preferable to dropping the charge in a mass from the top of the furnace. In

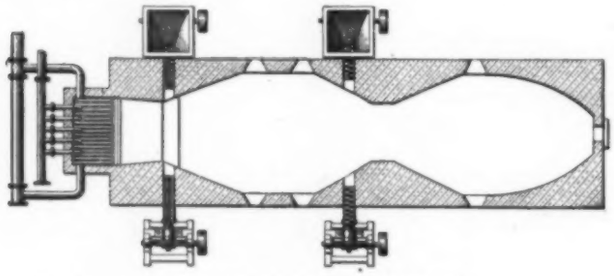


FIG. 5.—REVERBERATORY FURNACE.

applying the heat to the charge it is proposed to do away with the bridge, and direct the flame from the elevated firebox, downward on the hearth of the furnace against the charge, by means of a series of jets of air or steam. (978,621, Dec. 13, 1910.)

Baggaley Process of Smelting.—Metallurgists will recall the experimental work of Ralph Baggaley, of Pittsburg, Pa., as carried on at the smelting works of the Pittsburg and Montana company at Butte, Mont. It was his intention to provide an inexpensive means for the direct treatment of gold, silver and copper ores, without preliminary concentration or other treatment, and thus avoid the losses incident thereto. Where such ores contained sufficient fuel to provide heat for smelting he added no carbonaceous fuel. His furnace was a converter-smelter, for the production of a low grade matte, which he afterwards enriched and converted by treatment with silicious ores, after the manner of the present practice of basic converting. The process was in operation for several months, but later abandoned. Metallurgists interested in knowing the details of Mr. Baggaley's ideas may find a very complete description in his latest patent specifications, numbers of which are given herewith. (977,922, 977,996, Dec. 6, 1910.)

Zinc.

Volatilization Process and Furnace.—An improvement in the metallurgy of zinc is the basis of a patent granted to Harry H. Hughes, of Springfield, Mo. In our issue for August, 1910, page 502, we gave some details of the furnace, which consists of a revolving cylindrical retort and condenser, with provision for keeping air from entering the apparatus. The feed is continuous, and spelter and residue are discharged separately. In a more recent patent granted to the same inventor, he provides a larger and stationary condenser in place of the small rotating one indicated in his previous patents. (980,640, Jan. 3, 1911.)

Furnaces.

A stirring device for mechanically rabbled furnaces has been patented by Utley Weige, of Ardmore, Pa. The illustration Fig. 6 represents a vertical cross section of the improved rabble arm and rabble. The stirrer arm is represented by 1^a, the rabble head by 3^x and the rabble blade by 2. The rabble head rests on the stirrer arm, and is held in position while moving by means of the bent portion 4. The rabble head may rise to ride over any obstruction on the furnace hearth, the pivotal axis being near the point 4. Rabble arms may be interlocked on the arm by means of a tongue projecting from one into a recess of the other, as at 6. The rabble blades are made independent of the head, so that they may be removed when worn. They fit onto the head with a dovetailed joint as shown. In order to prevent the corrosion of the stirrer arm it is provided with sleeves of refractory material as shown at 11. The advantages claimed by the inventor are the ease of removing any rabble head or blade, and the reduction in cost of the parts and of the roasting operation. Alternate forms of the interlocking rabble head, and of the method of applying the refractory sleeve, are shown in the specifications. (980,717, Jan. 3, 1911.)

Dust prevention in the discharge of roasted ore from a furnace, is the object of a device patented by Thomas Edwards, of Ballarat, Australia. The machine consists of two or more cylindrical chambers mounted one above the other. A central rotary spindle extends vertically through these chambers, and carries a number of apertured discs which rotate in the bottom of each chamber. As the discs revolve they periodically register with openings in the bottom of each chamber and allow ore to pass to the chamber below; but the discs are so arranged on the spindle that the apertures in all the chambers are never open at the same time, so that the ore is discharged from the furnace in small quantities by a series of short drops. This prevents the dusting which is inevitable when large quantities of hot are discharged from a furnace. If desired the chambers may be water-cooled, and the ore thereby cooled for a subsequent operation. (979,937, Dec. 27, 1910.)

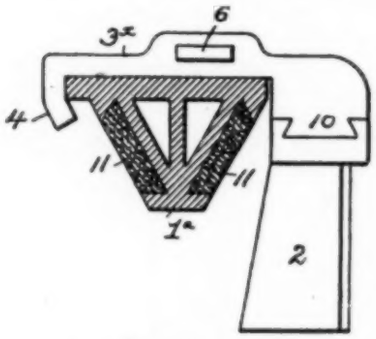


FIG. 6.—STIRRING DEVICE.

Granulating Slag.

A process for granulating molten slag recently has been patented in the United States by Heinrich Colloseus, of Berlin, Germany. It already has been proposed to granulate slag by allowing it to flow over a rotating drum having nozzles in its outer circumference through which liquid could be ejected for the purpose of assisting the granulating process, or for adding a desirable substance to the slag. The present inventor finds that the process is greatly facilitated by directing horizontal streams of fluid just beneath the flowing stream of slag and just as it falls on the revolving drum. (980,606, Jan. 3, 1911.)

Electrolytic Processes.

Chlorates and Hypochlorites.—In the production of chlorates and hypochlorites by chloride electrolysis without a diaphragm (as well as in other electrolytic processes) it is of importance to prevent the hydrogen bubbles set free at the cathode from sticking to it. For this reason Mr. Frank H. Briggs of Elyria, Ohio, does not use a straight plate for cathode, but forms it with slanting sides so that the hydrogen bubbles detach and rise. For chlorate and hypochlorite production he uses a lead cathode coated with platinum. (981,371, Jan. 10, 1911.)

Persulphuric Acid.—In the past the production of persulphuric acid by electrolysis of sulphuric acid has been carried out by cooling the electrolyte below 10° C. and adding some hydrochloric acid to increase the yield. Dr. G. Teichner of Nürnberg, Germany, has discovered that no cooling of the whole electrolyte is necessary and that the addition of hydrochloric acid may be omitted, if only the anode is maintained at about 15° C. The electrolyte may be at a temperature of more than 20° C. He makes the anode hollow and passes water of 12° C. to 15° C. through it. He uses sulphuric acid of specific gravity 1.35, an anode of platinum, a cathode of lead and an anodic current density of 0.5 amp. per square centimeter. The electrolysis is carried out as rapidly as possible to avoid

the formation of Caro's acid which has a depolarizing action. This is effected by making the volume of the electrolyzed liquid as small as possible, for instance, 5 cu. cm to 10 cu. cm electrolyte for 1 amp. (981,900, Jan. 17, 1911.)

Cleaning Metallic Articles.—To clean articles of gold, silver and other metals from the tarnish of oxide and the like, Mr. H. E. GENET of New York City immerses the articles in a solution of hot sodium hydroxide and brings them in contact with a piece of aluminium, for instance, in form of a perforated aluminium dish. This represents a short-circuited galvanic couple and the oxide tarnish is reduced to metal. (980,627, Jan. 3, 1911.)

Mercury from Poor Ores.—To extract mercury from poor ores, Mr. BELA SZILARD of Paris, France, uses as lixiviating agent a mixture consisting of a hot solution of sodium chloride and of calcium sulphide obtained as a by-product in the manufacture of soda by the Leblanc process, or of other calcium sulphides rendered alkaline by oxide of calcium. The tepid mixture is added to the finely crushed ores, and after standing for two days a current of hot air is passed through the mixture from below. The calcium sulphide is converted into calcium thiosulphate which is deposited with the ores. The liquid discharged contains mercury not in the form of a sulphide but in the form of a double salt solution of mercury and sodium chloride which is submitted to electrolysis. Any silver contained in the ore is also dissolved and precipitated by electrolysis. The lixiviation is best carried out in an autoclave boiler at a pressure of four atmospheres. (980,364, Jan. 3, 1911.)

Electric Furnaces.

Ferroboron.—To produce carbon-free ferroboron for use in the steel industry, Mr. C. A. HANSEN of the General Electric Company, Schenectady, N. Y., uses silicon as reducing agent. The mineral colemanite $Ca_2B_6O_{11}.5H_2O$ is first dehydrated by heating to a temperature of 450° C. It is then powdered and mixed with powdered ferrosilicon (since this is cheaper than pure metallic silicon). Sufficient silicon must be added to reduce all of the boron. The powdered mass is mixed with a suitable binder, like waterglass, and pressed into pellets. The pellets are heated in an electric furnace to a temperature above 2000° C. The silicon reduces the boron and is itself oxidized to silica which combines with the calcium to form a slag of calcium silicate which separates perfectly from the metal matrix. (982,135, Jan. 17, 1911.)

Pure Metals.—Pure tungsten, tantalum, vanadium, etc., are produced by Mr. WM. C. ARSEM, of the General Electric Company, by reduction in his vacuum tube furnace. The reducing agent is very finely divided carbon, produced from lampblack by heating to a very high temperature in a vacuum or by graphitizing. (979,363, Dec. 20, 1910.)

Carbon Electrodes.—In the manufacture of large carbon electrodes, Mr. WALTER GASTON of New York City proceeds as follows to prevent them from cracking. The electrode as it comes from the baking furnace hot is plunged into boiling tar and pitch so as to impregnate it thoroughly. The pores are thereby filled. The operation really adds a second binder to the electrode. The absorption of moisture during storage and transportation is thereby avoided. The addition is stated not to impair the conductivity, but rather to improve it. (979,465, Dec. 27, 1910.)

Carbon Electrode.—In the manufacture of very large amorphous carbon electrodes, such as are now used in electric steel furnaces, considerable difficulties are experienced in baking them uniformly. Mr. RAPHAEL H. WOLFF, of New York City, proposes to overcome this difficulty by making the electrode essentially of two pieces, an exterior and an interior. The exterior part is hollow, for instance, a cylinder, the inner hole being wider at one end than at the other. This hole is then filled either with one solid piece or a combination of pieces, and tar pitch is infused to bind the whole together. (974,008, Oct. 25, 1910.)

Recent Progress in the Electrometallurgy of Iron and Steel.

A two-ton Roechling-Rodenhauser electric combination induction furnace is to be erected by the American Electric Furnace Company of New York some time during this year at Lansdowne, Pa. This is the first furnace of this type to be installed in this country.

The Girod furnace installed by the Simonds Manufacturing Company at Chicago is now being transferred to their new and modern plant at Lockport, N. Y., and will be used for producing and refining the high-grade steels required for their saws, machine knives, files, etc.

Interesting information comes in reliable private reports from Europe. Amorphous carbon electrodes are there in successful use in steel furnaces in sizes up to 600 mm (24 in.) diameter. In commercial practice with European Heroult furnaces the consumption of these carbon electrodes is from 3 kg to 4 kg per ton of steel. A new 20-ton to 22-ton Heroult furnace—the largest in size so far—is soon to be erected in a German works. In another German works using the Heroult furnace (seven tons capacity) the energy consumption (with fluid charge) is given as 105 kw-hours per ton of steel.

From Messrs. C. W. Leavitt & Company of New York, the American representatives of the Girod furnace, we have received the following data on the experience of the Cockerill Company in Seraing, Belgium, who have been operating a 4-ton to 5-ton Girod furnace for the past two years and are refining molten Thomas steel for rails, locomotive tires, cannons, automobiles, etc. The total cost of power, labor, furnace upkeep, electrodes and general expenses for the operation of this electric furnace is given as less than $4. The Krupp firm in Essen is installing a 12½-ton Girod furnace to be operated by three-phase currents.

* * * *

As to the reduction of iron ore in the electric furnace interesting information comes from Scandinavia.

The large 2500-hp electric shaft furnace at Trollhättan was put in operation in November and appears to work perfectly well. The furnace was first tapped on Nov. 16, and from a private report written two weeks later we learn that the furnace had been working without difficulty since the start, and especially that none of the electrodes had been touched in that time.

From another report, dated Dec. 10, it is learned that the furnace had been running continuously since Nov. 15 and was working quite satisfactorily.

The furnace plant is equipped with apparatus for making gas analyses and taking temperatures at every 3 ft. of the whole stock. There is also a very elaborate laboratory for testing the raw materials and the product obtained

The yield was for the first week 2990 kg (6578 lb.) per hp-year, but has since been raised to above three metric tons (about 6600 lb.) per hp-year.

The consumption of coal amounted to 24 hectoliters to 26 hectoliters (85 cu. ft. to 92 cu. ft.) per 1000 kg (2204 lb.) of iron.

The works are not permitted to be examined by outsiders at present. But all on the spot, even previous enemies of the system, are reported to be enthusiastic over the undertaking.

At Domnarpoet, when the first electric iron-ore smelting experiments by this system took place, a new furnace for from 4000 hp to 5000 hp is at present under erection. It is expected to be ready for operation in May.

At Tyssedal, Hardangen, Norway, there is an installation of two electric shaft furnaces, each for 3500 hp, one of which will be put in commission next summer.

The Aktiebolaget Elektrometall, in addition to the foregoing, has given licenses to several other works, one of which is the Uddeholms Iron Works in Sweden, for which they are now preparing drawings for three furnaces of 3000 hp each.

Heating Liquids by Direct Steam.

In this article a description will be given of some appliances, made by the Schutte & Koerting Company, of Philadelphia, for heating liquids by means of direct steam.

Fig. 1 shows a continuous water heating device. The steam enters at *S* and the water at *W*. Heating is effected by the mixture of steam and water taking place through nozzles in the instrument. *D* is the discharge.

This apparatus may also be used as a condenser and as an

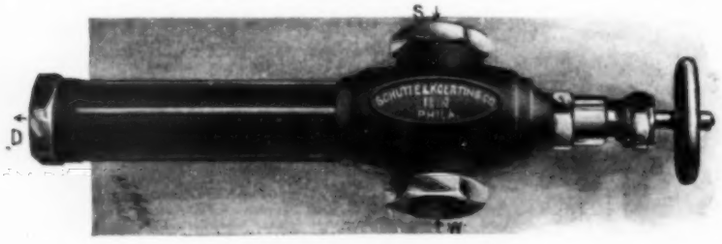


FIG. 1.—CONTINUOUS WATER HEATER.

exhauster for heating coils and on the exhaust pipe of engines. It can be used with steam of any pressure or with exhaust steam for the maintenance of a partial vacuum, or as an exhauster for the removal of air only. The capacity and range in volume is regulated by the central hand wheel, and when used for heating water the temperature is regulated by throttling with a valve in the steam pipe.

Fig. 2 shows the use of the attachment for the production of hot water. The water is supplied from the bottom, the steam from the top. Valve *B* is a stop valve in the water supply pipe, while the volume of the water is regulated by the spindle *A*. The temperature is regulated by throttling with valve *C* in the steam supply. This valve can also be used as a stop valve in the steam pipe, but it is more convenient to provide a separate lever valve *D* for turning on and shutting off the steam. To start, the water is first turned on at *B* and then the steam turned on at *D*. To stop, the steam is first shut off at *D* and then the water shut off at *B*.

Fig. 3 shows an attachment to an engine exhaust or heating coil. In this attachment it is necessary to have a check valve *E* to prevent a back flow of water, and also a free exhaust valve *F* is required. The discharge is run free to waste or all or part of it into a hot water tank for use.

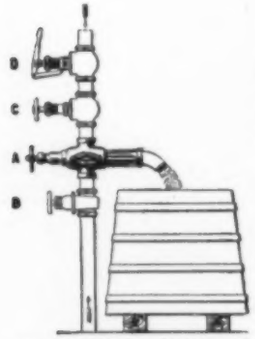


FIG. 2.—ATTACHMENT FOR PRODUCTION OF HOT WATER.

Fig. 4 shows the application of this continuous water heater for making warm water in different stories of buildings. The installation is very simple, *S* being the steam pipe, *P* the water pipe, *H* the heater.

The continuous water heater offers a means of heating the water exactly to the desired temperature, the regulation being obtained by the steam spindle.

The circulating heater, Fig. 5, does satisfactory work when water need not be heated above 160° and where steam pressure is not less than 35 lb. This circulating heater, within the above limits of temperature and pressure, works without noise or nearly so, and works very economically. *S* is the steam inlet, *D* the discharge of the hot water.

The disturbing noise and destructive effect on open tanks, when steam is introduced into the water through perforated pipes, is entirely obviated by attaching the noiseless heater to the end of the steam pipe. The heater discharges in the form of an inverted cone, offering a large surface and thus causing rapid condensation (Fig. 6).

If water is to be heated to a temperature not exceeding 165° Fahr. the air inlet on heater is closed with a plug and nothing but the heater is needed, as it will be noiseless with any pressure of steam from 5 lb. to 100 lb. and above. If water is to be heated to a temperature above 165° Fahr. and up to boiling point, then an air pipe must be attached. The purpose of this pipe is to permit air to be supplied to the discharge jet and by its admixture prevent the formation of steam bubbles which otherwise in rising to the surface would escape and collapse and thus cause the noise. Thus water is heated in less time and with less expenditure of steam than could be done by the use of a perforated pipe. The air is drawn in through the action of the heater if the steam pressure is proportionate to the depth of the water in the tank above the apparatus, viz.:

Height of water above heater or depth of tank in feet	3	4	5	6	7	8
Minimum steam pressure, in pounds	4	8	12	18	24	32

Thus under the above conditions the heater will work without noise up to any temperature by the addition of the air pipe, which is best fitted with a cock or valve. The heater works also with exhaust steam. Fig. 7 shows an installation.

Bureau of Standards Analyzed Samples.

The following announcement of the Bureau of Standards in Washington, D. C., will be of interest to chemists and metallurgists at large.

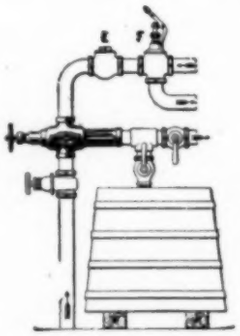


FIG. 3.—ATTACHMENT TO ENGINE EXHAUST.

The Bureau of Standards is ready to supply the following new analyzed samples:

No. 29, titaniferous magnetite ore from New Jersey.

No. 11a, basic open-hearth steel with 0.2 per cent carbon.

No. 12a, basic open-hearth steel with 0.4 per cent carbon.

The steels replace the original samples which are exhausted. Renewal samples of Bessemer steels with 0.1 per cent and 0.2 per cent carbon are in process of analysis, also a nickel and a chrome-vanadium steel.

The Glasgow Smoke-Abatement Exhibition.

By John B. C. Kershaw.

The Glasgow Smoke Abatement Exhibition, which was recently held in the Victoria Skating Rink in that city, has been an unqualified success, the attendance having averaged over 4000 persons per day, and a large amount of business having been obtained by the exhibitors as a direct outcome of their displays of smoke-abatement appliances.

The exhibition was promoted by the corporation of the city, and the three sections into which the exhibition was divided were under the charge of the leading officials connected with the gas, electricity and sanitary undertakings of the city. The success of the exhibition was no doubt largely due to the rivalry between the gas and electricity sections of the exhibition as to which should have the better display.

This combination of gas and electricity appliances under one roof is a new feature in smoke-abatement exhibitions, and the success of the Glasgow exhibition indicates that it is one worth imitating. There is no doubt that where, as in Glasgow, both the gas and electricity supply undertakings are owned and worked by the municipality a joint exhibition of cooking and

heating appliances can be easily arranged and can be made very attractive to the general public.

In their prospectus the promoters of the exhibition state "that for a considerable time the Corporation of Glasgow has been devoting much attention to the problem of air purification, recognizing how necessary an unpolluted atmosphere is to the mental and physical well-being of the community. Special committees have gone into the subject with extreme care—reports have been prepared by experts—and numerous experiments have been made and the results recorded. From these it is clear that the question of smoke abatement is a vital one in towns. Having regard to this fact the corporation has decided that it would be most desirable to bring before the citizens of Glasgow and the west of Scotland, in a practical and interesting fashion, the latest and most scientific devices in connection with smokeless heating and lighting, embracing the most up-to-date methods and contrivances for the complete combustion of fuel, as well as the most modern developments in heating, lighting and ventilation by means of gas and electricity. The readiest means of interesting the public in these matters is to hold an exhibition which will bring before them in a concrete form the advantages to be gained and the economies to be effected by the adoption of recent discoveries; and the corporation has accordingly entrusted to a special committee the organization of such an exhibition."

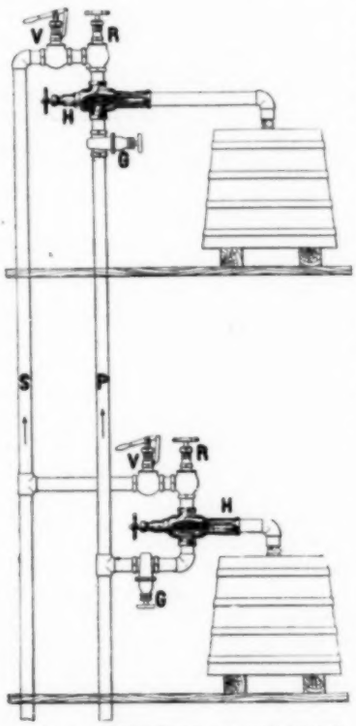


FIG. 4.—PRODUCTION OF HOT WATER IN DIFFERENT STORIES.

The appliances and materials exhibited at Glasgow related chiefly to the domestic smoke problem, and only a few of the exhibits dealt with the abatement of the smoke arising from the furnaces and steam boilers used in manufacturing operations.

FIG. 5.—CIRCULATING HEATER.

The exhibition was divided into three sections, the largest and most important being that devoted to the display of gas lighting, heating and cooking appliances. The second section was devoted to the use of electricity for similar purposes, and the third section was given up to exhibits of smokeless fuels and of patent stoves and grates.

The most important and striking exhibits in the first two sections were the model houses in which every lighting, heating and cooking appliance was operated in the first case by gas and in the second case by electricity.

In some of the smaller Scotch towns the gas committees are now installing gas grills, stoves and meters free of charge, it having been found that this free gift of gas heating appliances pays the corporations by the increased sale of gas. The gas committee of the Glasgow corporation is contemplating a similar step and when this is taken the electricity committee is almost certain to follow in their footsteps. With electricity at 2 cents per kw-hour and gas at 50 cents per 1000 cu. ft. the cost of lighting and heating one's house by either gas or electricity in these Scotch towns need not be prohibitive; and when the public has become accustomed to the comfort and convenience of the newer methods of lighting and heating the days of the coal-fired open grates will be numbered. Already there are stated to be 80,000 gas cooking appliances in use in Glasgow alone, and their use is extending.

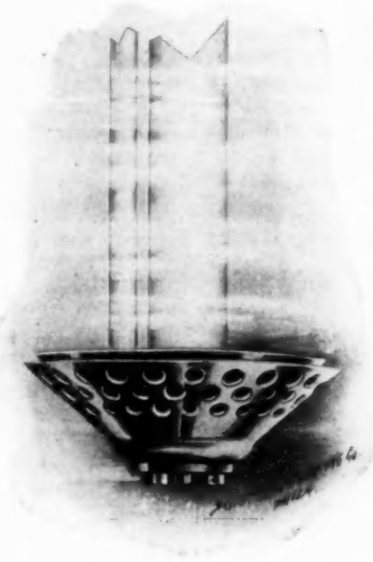


FIG. 6.—CONE DISCHARGE.

In Section II of the exhibition a wonderfully varied and attractive display of electric lamps and heating appliances was shown, and here again it would be invidious to single out any special firm's display for mention in this report. The exhibit of electric radiators and of cooking appliances was probably the best that has yet been collected in the United Kingdom and there is no doubt that the Glasgow Electricity Undertaking will have a largely increased day load as a result of the 1910 exhibition. It may be stated here that in Glasgow it is proposed to supply electricity for heating from the lighting circuits, the amount consumed being calculated from the total current registered by the meter, and the average lighting consumption in previous quarters. All above the average lighting consumption is charged at 2 cents per kw-hour and is assumed to have been used for heating purposes. By this system the expense of separate wiring and meters is avoided.

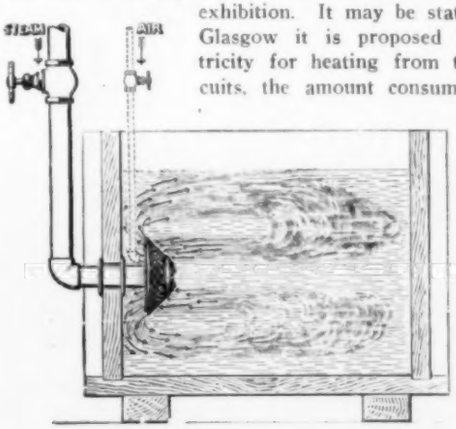


FIG. 7.—CONE DISCHARGE INSTALLATION.

The smokeless fuel and patent grate and stove section of the Glasgow exhibition was the smallest of the three and attracted less public attention than the other two for reasons which are obvious, since electricity and gas lend themselves in numberless ways to the creation of striking and attractive exhibits. The smokeless fuel exhibits comprised peat from the

Glasgow distress committee's works at Palacerigg; anthracite from the Scotch and South Wales coal fields; coke, "charco," "coalseld" and "coalite."

The three last are patented smokeless fuels, with a limited sale in the United Kingdom.

"Charco" is made by subjecting ordinary coke or partially carbonized coal to a special treatment before cooling, as it comes from the gas retorts or coke ovens. It ignites readily, burns steadily and gives out greater radiant heat than either coal or coke, while being absolutely smokeless.

"Coalseld" is a similar fuel, produced from ordinary gas-coal by use of a modified form of gas retort. It is now being made at the Glasgow Corporation Gas Works, and sample orders are being supplied by the gas department of the city.

"Coalite" is coal carbonized at a low temperature (700° Fahr.) and supplied by the British Coalite Company, whose main works are at Barking, near London. It is unfortunate that the company producing coalite has not so far been a financial success, and that in the north of England it is practically impossible to purchase this fuel for trial purposes.

The improved grates and stoves at the Glasgow exhibition were chiefly of the well-known and widely used types, the only novelties being open grates fired on the under-feed principle, which has been successfully used for boiler firing. These grates have been placed on the market by the White Flame Fire Company, of Stirling, and depend for their action upon the partial coking of the raw coal before it reaches the well of the grate. A hopper at the back of the fire contains the day's supply of fuel; this hopper is closed with a sliding gas-tight cover.

The coal is heated and partially coked in this hopper, and the hydrocarbon gases can find no outlet except through the opening into the grate, where they are consumed as they pass through the glowing fuel. The solid fuel is raked forward through the opening of the hopper as required to replenish the fire.

The makers of this grate do not claim that it is entirely smokeless, since when the fire is first lighted it labors under the disadvantages of the ordinary open fire as regards imperfect combustion and smoke production. But when the fuel in the grate and the hopper behind it have once attained their maximum temperature the fuel is smokelessly consumed, while all the advantages of the ordinary open fire, good ventilation and cheerfulness, are retained.

The show of smokeless kitchen stoves for use with ordinary coal was also a good one. In these the conditions required for smokelessness are more easily attained, since most cooking stoves are now designed on the closed-in principle and only require scientific design to obtain perfect combustion of the fuel. In one of the kitchen stoves exhibited at Glasgow the combustion chamber of the stove is extended under the hot-water boiler, and the hot gases then pass under the hot plate and oven before escaping to the chimney. By this arrangement it is claimed that double the amount of hot water is obtained, with a saving of 50 per cent in fuel.

The use of glass for the doors of ovens and plate-warming chambers is another novelty found in some of the kitchen stoves exhibited by J. C. Parker & Company, of Glasgow, and attracting some attention.

A large number of anthracite and coke stoves were exhibited by various well-known makers, and there is no doubt that the popularity of the former for heating offices and rooms is widely extending, since they require little attention and are most efficient heating appliances.

The Glasgow exhibition, although confined to only one branch of the problem of smoke abatement, namely, that of "domestic smoke," has been, as regards attendance and results, more successful than the Smoke Abatement Exhibition held at Sheffield early in 1909, and its organization and management will probably prove the model for the succeeding exhibitions of this character which are certain to follow.

NOTES.

Chemists' Club.—The new home of the Chemists' Club is making very satisfactory progress and is rapidly nearing completion. The laboratory floors and the public stairway will be completed by February 1. Over half of the available laboratory and office space has already been rented.

The American Rolling Mill Company, of Middletown, Ohio, has just completed a splendidly equipped research laboratory costing approximately $40,000. The excellent organization which the company is known to have, has recently been strengthened by securing the co-operation of Dr. A. S. Cushman, of Washington.

The Hardinge Conical Mill Company have removed their offices from 35 Wall Street to 50 Church Street, New York City.

New Electrolytic Furnace Process for Metal Production.—On page 688 of our December issue mention was made of a patent of Mr. J. W. Beckman, of Niagara Falls, Ontario, for producing pure carbon-free metals and their alloys from their oxides and sulphides. The process is applicable to a great number of metals, since only those are excluded which are more electropositive. We learn that among other metals, vanadium and titanium have been produced by this process on a laboratory scale. The process depends upon the fusibility of a salt composed of CaO and the oxide of the metal to be produced. This fused salt provides a cheap electrolyte in which the oxide is dissolved and electrolyzed. On account of the high boiling point of this electrolyte, the metals, even those with a comparatively high melting point, can be deposited in a molten condition and the operation is very simple on this account.

The Stuart & Peterson Company, of Burlington, N. J., have sent us their new catalogue No. 219, which contains on 176 pages illustrations and concise descriptions of the many products of their manufacture, including plain and porcelain-lined ware, steam-jacketed kettles, hollowware, chemists' appliances and specialties, furnaces, heaters, vacuum pans, retorts, tanks, etc.

Calendars.—The Goldschmidt Thermit Company, of New York City, have sent us their pretty calendar for 1911 with a map of the United States and some interesting illustrations of thermit welds. The attractive calendar of the Lead Lined Iron Pipe Company, of Wakefield, Mass., has a fine picture of the new Pennsylvania Railroad Terminal in New York City, which contains over 100,000 feet of lead-lined iron pipe for hot and cold water, made by that company.

The Arizona Copper Company produced for the month of November, 1910, 1505 tons of copper.

The Bureau of Mines, Washington, D. C., has recently published two bulletins, Nos. 2 and 3, of the new series. They deal respectively with North Dakota Lignite as a Fuel for Power Plant Boilers, and The Coke Industry of the United States as related to the Foundry.

Bakelite.—Information No. 1, issued by the General Bakelite Company, 100 William Street, New York City, gives first a brief outline of what bakelite is and of the properties of bakelite C and liquid bakelite A. Then follow brief chapters on bakelite lacquer, bakelite protective enamels (for the protection of metallic surfaces, pipes and machinery against corrosion), bakelite-impregnated wood, bakelite adhesive cement (for joining iron, wood or other hard materials with a cement, which, after hardening, can withstand moisture, steam, alkalies, acids, solvents and heat), bakelite plastic mixtures, bakelite paper and bakelite-asbestos paper. The pamphlet, which is exceedingly neat, is concluded by directions for molding bakelite paper and other bakelite plastic mixtures and by a description of the bakelizer which is the apparatus for heating under increased atmospheric pressure to convert bakelite A and B into bakelite C.

Hardening and Annealing.—We have received from Mr.

C. U. Scott, of Davenport, Ia., his illustrated catalog on gas-blast furnaces, fuel gas plants, hardening furnaces for high-speed tool steels, tempering furnaces, annealing furnaces, etc. Mr. Scott acts as a consulting engineer for the heat treatment and hardening of steel, besides the manufacturer of furnaces and gas plant for this purpose. This firm also undertakes hardening of tools made from carbon or high-speed steels, also case hardening and heat treatment of steel in general.

Messrs. **Merck & Co.,** manufacturing chemists, of New York City, with works at St. Louis, Mo., and Rahway, N. J., announce the removal of their headquarters to 45 and 47 Park Place, New York City, on Jan. 1, 1911.

A Flux for Brazing Metals, notably cast iron, is composed of potassium cyanide, one part by weight, and borax glas, two parts. The pieces to be brazed are heated to redness, when the flux is applied and the parts immediately become cleansed and reduced. A brazing solder, such as Hungerford's No. 24, may then be applied with success. The flux mentioned was made the subject of a patent by W. W. McIver, of New York.

The **T. L. Smith Company,** Milwaukee, has issued a new catalog describing the Symons crusher. The Symons is a gyratory which is lighter and more compact than the usual form of this machine. The upright shaft is stationary, and the motion is imparted to the crusher head by means of an eccentric sleeve which revolves on the shaft.

Symons Brothers, Chicago, have recently designed a new disc crusher, not a grinder, which is said to have extraordinary capacity. It reduces rock or ore to ⅛ or ¼ in. and is designed to have a place in stage crushing. The manufacturers have issued a descriptive bulletin.

Camp Bird, Ltd., owning the Camp Bird mine at Ouray, Col., and the Santa Gertrudis property in Mexico, is prosecuting work at the latter in anticipation of the forced abandonment of the former within a year or so on account of exhaustion of ore. Developments at the Santa Gertrudis indicate that a modern mill will be erected, comprising stamps, tube-mills, cyanide tanks, filter presses, precipitation presses and appurtenant machinery. The plant will have a capacity of 600 tons per day. The November report of the company shows that the Camp Bird mill crushed 6577 tons of ore, producing 6218 oz. of bullion and approximately 723 tons of concentrate. The balance applicable to profit was $98,500.

The Arizona Copper Co. produced in the month of December, 1910, 1612 tons of copper.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in October, 1910, was 71. The total tonnage milled during that period was 1,916,801 tons of ore; 9756 stamps were in operation with an average duty of 7.31 tons per 24 hours. Tube mills in commission numbered 188. The yield for the month was 653,147 fine ounces gold. The statement shows a great advance over the same period for last year.

Evaporator Patent Suit.—The patent suit of the Kestner Evaporator Company versus the American Evaporator Company has recently been decided in favor of the defendant. The decision is written by Judge Holland. Infringement was claimed of patent 548986 of Oct. 28, 1895, of Peter Cooper Hewitt; the claim in question reads as follows: "In an evaporator provided with heating tubes and one or more return tubes, a separating chamber provided with a series of stationary tangential nozzles communicating with the heating tubes, substantially as specified." In the defendant's evaporator a film of liquid travels along the inner surface of the heating tubes under the influence of a vacuum to an upper chamber where the vapor is given a circular motion and the particles of liquid separated from the vapor by centrifugal force, after which the vapor passes off. But since the liquid is not returned for retreatment through "one or more return tubes," as the liquid passes once through each effect only, the above mentioned claim is not found to be infringed.

Gold pannings obtained in the field may be preserved for assay by using lead foil, 4 in. square, into which the values are washed after panning. The foil is bent into basin shape, and after receiving the washings, which are drained off, may be rolled up and marked with a number or letter. The gold is then in shape for cupellation, if very clean, or for fusion if mixed with much silica. The process avoids the possibility of loss or salting, and is suggested by Mr. J. E. Wood in *Bull.*, No. 75, I. M. M.

Solubility of Oxygen in Molten Silver.—Spitting and foaming of molten silver at the time of solidification and blistering of solid silver when it is rolled are due to dissolved oxygen. F. G. Donnan and T. W. A. Shaw recently determined the solubility of oxygen in molten silver and presented their results to the Soc. of Chem. Ind. They devised special apparatus and ascertained that the volume of oxygen (measured at 0 mm and 760 mm) dissolved in 10 g melted silver at 1020° C. and 751 mm pressure is 20.5 ccm. To combine with this quantity of oxygen 0.03 g of aluminium would be required to be mixed with each 10 g silver.

The Sulitelma Company, operating in Norwat, produced in the month of September 760 tons of copper concentrate by the Elmore process.

La Rose Consolidated Mines Co., operating in the Cobalt district, Canada, gives the following information relative to the milling of its low-grade ore, covering the period from Sept. 8, 1909, to May 31, 1910:

Average assay crude ore	28.58 oz. per ton.
Average assay tailing	3.68 oz. per ton.
Tonnage treated	18,423.56 dry tons.
Concentrate produced	628.02 dry tons.
Contents of crude ore	526,581.00 oz.
Contents of concentrate	416,629.59 oz.
Average assay of concentrate	663.00 oz. per ton.
Actual extraction	79.12 per cent.
Ratio of concentration	29.3 to 1
Net value of concentrate	$199,569.70
Cost of concentration	$61,351.54
Profit on low-grade ore	$138,218.16

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines for the month of August, 1910, was 72. Tonnage milled, 1,884,288; number of stamps dropping, 9685, with an average duty of 7.15 tons per 24 hours; tube mills in commission, 182. Comparison with last year shows fewer stamps and more tube mills at work, a larger tonnage treated, increased working cost per ton milled and decreased working profit. The yield for the month was 649,269 fine ounces, being larger than for any month in 1909 or 1910.

The Utah Copper Co. has issued its tenth quarterly report covering the third quarter of 1910. During July all three of the concentrating plants were in operation, but at the end of that month the Copperton plant, at the mine, was closed permanently and dismantled. The equipment is being used to remodel the Arthur plant at Garfield. The improvements at the Magna plant are completed and that plant is now treating in excess of 10,000 tons per day. When the Arthur plant is remodeled on the plans corresponding to those of the Magna plant it will have a capacity of about 6000 tons. The tonnage milled for the quarter was less than that for the preceding quarter, on account of the changes being made in plants as indicated. The production was 23,194,921 lb. of copper as compared with 25,124,052 lb. for the second quarter. The total net profit for the quarter was $1,329,966.88, from which a dividend of $1,168,882.50 was paid. The average cost per pound of copper produced, after smelter deductions were made, was 8.25c. as compared with 7.53c. for the preceding quarter. The increased cost of production is ascribed to the decreased tonnage treated. The average copper content of the ore was 1.5 per cent for the quarter and 1.4 per cent for the month of September.

Personal

Mr. **W. H. Aldrich,** late managing director of the Consolidated Smelting & Mining Company, of Canada, is to have a voice in the management of two of the large Western copper companies, the Inspiration in Arizona and the Mason Valley in Nevada. His headquarters will be at Los Angeles.

Mr. **A. J. Bone** has been appointed smelter superintendent of the Copperhill smelter of the Tennessee Copper Company.

Dr. **Regis Chauvenet,** formerly president of the Colorado School of Mines, is at Mount Franklin, near El Paso, Tex., where he is engaged in tin mining and smelting.

Mr. **N. H. Darton** has been appointed geologist for the new Bureau of Mines. He has been connected with the United States Geological Survey for many years.

Mr. **Forrest Goody** has been in Denver for some weeks preparing plans and specifications for a metallurgical enterprise in Nevada.

Mr. **G. W. Heintz,** general manager of the United States Smelting, Refining & Mining Company, Salt Lake City, recently returned from a trip of inspection of the new baghouse system Company of Canada, with headquarters at Trail, B. C.

Mr. **Victor G. Hills,** Denver, investigated some tungsten deposits in Nova Scotia last month.

Mr. **Martin E. Hiltner,** Deadwood, S. D., is to have charge of the metallurgical opeerations of the New Reliance Mining Company, of South Dakota. He was recently in Denver purchasing supplies for the mill.

Mr. **Ernst Leitz,** of Wetzlar, Germany, the famous manufacturer of microscopes, etc., has received from the University of Marburg the honorary degree of Doctor of Philosophy, in recognition of his services rendered to science by having constructed for more than fifty years instruments of highest scientific value.

Mr. **Jay Lonergan,** Colorado School of Mines, '05, has accepted a professorship in the Imperial University, Tientsin, China, and has left Denver for his new field.

Mr. **R. H. Stewart** will succeed Mr. W. H. Aldrich as managing director of the Consolidated Smelting & Mining Company of Canada, with headquarters at Trail, B. C.

Mr. **A. H. Wethey,** who for many years has been in charge of Senator Clark's interests in Butte, retired from his position on Jan. 1. Senator Clark has disposed of so many of his former interests that the retention of a general manager is unnecessary.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ELECTRIC FURNACES (*Continued.*)

698,980 and 698,981, April 29, 1902, Johann Lühne, of Aachen, Germany.

Combined arc and resistance type. Material for making glass is melted by electric arcs and further heated by solid resistors. The melting apparatus is a trough-shaped receptacle having two flat vertically grooved side walls converging downward, jacketed with kieselguhr, ashes or glass wool. The negative electrode is a horizontal carbon bar, square in cross-section, extending from end to end of the trough near its bottom. It is hollow and perforated, and hydrogen, or a mixture of hydrogen and air, is supplied through it, being burned in the furnace to assist in melting the material. A series of horizontal positive carbon electrodes passes through each side of the melting chamber, their ends being in proximity to the angular sides of the negative electrode, and arcs being sprung between them and the negative electrode. The molten product passes down through a valved opening onto an inclined hearth. This hearth either contains resistors consisting of solid electrolytes, e. g., oxides of beryllium, calcium, zinc or titanium, or is composed of such materials containing embedded metal terminals. No current passes through these resistors, or resistor hearth, until they are heated by the descending molten material, or by hot air. As the molten glass passes over this hearth the light impurities derived from the electrodes are removed by a skimmer, or retained by a depending partition. The glass thence passes through a series of concentric annular chambers, partially separated by partitions or dams.

699,333, May 6, 1902, A. Ludwig, of Bernburg, Germany.

Arc type, for heating materials in an atmosphere of highly compressed hydrogen or other gas, e. g., for the production of artificial diamonds. The chamber is a thick metal tube, closed at the lower end and surrounded by superposed flat metal rings. The cover is a massive plate of metal, secured on the open end of the cylinder by a metal strap consisting of a series of plates bolted together. Opposed wedges are driven between the strap and cover. For very high pressures the strap is heated before the wedges are tightened. Th wedges may have division lines to regulate their position and the pressure of the cover on the cylinder. The gas pressure may be several thousand atmospheres. The electrodes are vertical carbon rods, one extending up from the bottom of the cylinder and the other depending and adjustably carried by a revoluble spindle journaled in the cover. A gas inlet pipe also passes through the cover. The spindle and pipe may be held by wedges.

699,654, May 13, 1902, Guillaume DeChalmot, of Holcomb's Rock, Va. Assignor to the Willson Aluminum Company.

Arc or resistance type, for reducing ores, or producing carbides or silicides. The furnace is a pot of iron lined with carbon, consisting of a lower and an upper section. It is mounted on a truck. The lower section has a tap hole for metal and may have one for slag. The upper section has a gas outlet. The pot constitutes one electrode; the other one is a depending carbon bar or group of bars. The furnace has a flat top made of closely interfitting hollow sections of cast iron, cooled by water circulated through them. Several electrodes may pass through openings in the cover. The electrode or electrodes and cover are maintained at the same potential by an electrical connection. The electrodes pass through stuffing boxes consisting of plates adjustably secured to the cover, with a packing of asbestos fiber. The cover is insulated from the pot by a 2-in. luting of brick, with or without clay or mortar. After the run the product or slag may be solidified in the pot and removed in one piece by first lifting off the upper section. Each electrode may carry 1000 or more amperes and may be independently switched into or out of circuit.

700,808, May 27, 1902, William R. Parks, of Chicago, Ill.

Arc type, especially for the production of calcium carbide. The furnace is a pot with brick sides. The lower negative electrode is a slab of carbon set in the floor of the pot, having a circular gutter leading to a central opening and drain. The positive electrode is a tube, vertically slitted and depending into proximity to the carbon hearth. The charge materials are delivered by a hopper and conveyer into the closed upper end of this electrode, and may be preheated therein by gas injected by pipes extending through the side of the electrode.

701,218, May 27, 1902, Fausto Morani, of Rome, Italy.

The furnace is a shallow pot, receiving a depending electrode. The pot has a water-cooled metal base, or imbedded water pipes. The depending electrode is a square bar, or

group of bars, carried by water-cooled metal yokes. The terminals are also water-cooled and are clamped against flat faces of the bars. The taphole is a slit in a hollow water-cooled casing of steel. The product is tapped into a water-cooled metal pan. May employ several electrodes supplied by polyphase current.

701,986, June 10, 1902, E. G. Acheson, of Buffalo, New York.

Eelectrode and terminal for furnaces, consisting, as shown, of 72 horizontal square carbon bars, piled in contact, with a copper plug screwed into the end of each bar. A vertical copper plate is bolted to these plugs and has a marginal flange with openings, into which are soldered the terminals of the flexible cables, preferably of the same number as the carbon bars. Seventy-two graphitized-carbon electrodes, each 4-in. square, will carry a current of 37,500 amp. A receptacle to contain water is bolted to the plate flange, and is connected to an insulated reservoir.

702,081, June 10, 1902, A. Voelker, of Ehrenfeld, Germany.

Resistance type. Material for the manufacture of glass is intimately mixed with graphite or powdered charcoal, about 10 per cent, to give a conductive mass which serves as a resistor. Saltpeter is also added to eliminate the carbon as carbon dioxide. Pure carbon is necessary, and the ash slightly colors the glass. The carbon may be replaced by conductive oxides or metals, e. g., "brownstone," or powdered iron or silver. These are only suitable for the production of colored glass. A continuous or alternating electric current may be used.

BOOK REVIEWS.

Metallography. By **Cecil H. Desch**, D. Sc., Ph. D., lecturer in metallurgical chemistry in the University of Glasgow. 429 pages, 14 plates, 108 diagrams. Price, $3. New York: Longmans, Green & Company.

This is probably the most satisfactory book yet written on this subject in English. The treatment throughout is clear and free from the mysterious phrases of other treatises. It is comprehensible from beginning to end by any bright, industrious person, and will give a clear understanding and good foundation in the subject. Numerous references to original literature are given; the microphotographs are excellent. It is really a treatise on the physical and chemical properties of metals and alloys.

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Elementi di Metallurgia. I: Siderurgia. By **Alfonso Greco.** 192 pages, 27 illus. Price, 1.50 lire ($0.45 retail price in New York). (Students' Library Series.) Livorno (Leghorn): Raffaello Giusti.

The characteristics of almost all Italian books, aside from their contents, are clear type, good paper, and wonderful cheapness. The equivalent of such a book as this in English would not be sold for less than $1. The metallurgy of iron as here set forth is elementary and slightly antiquated, about the equivalent of Bauerman's small English text-book, edition of 10 years ago. It can do good service in the Italian technical schools, however, for giving students their first ideas of the subject.

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An Introduction to the Study of Metallurgy. By Sir **W. C. Roberts-Austen.** Sixth edition, revised and enlarged by **F. W. Harbord.** 478 pages, 189 illustrations. Price, $6 net. Philadelphia: J. B. Lippincott Company.

Mr. Harbord has given this well-known text-book a new revision and has improved it in many respects. His revision of Chapter III—the physical properties of the metals—is not altogether satisfactory, much valuable fundamental information being still omitted, such as many of the boiling points, all the latent heats of fusion and latent heats of vaporization. The claim that Siemens was the progenitor of the electrolytic furnace (page 111) is very far-fetched, and detracts from the reliability of the work as a whole. The chapters on pyrometry and metallography are practically new and well done. The chapter on fuels has been much enlarged on the descriptive side, but on the side of calorific powers and calorific intensities it is still as wofully deficient as in former editions, viz., at least 20 years antique. The chapter on thermochemistry of metallurgical reactions has been improved a little, yet it needed a great deal more.

Altogether, the book is now a mixture of good, fairly good, and pretty bad; brought up to date in spots and back in the last century in others. We confess to being disappointed with the general effect; we had hoped that the revision would be altogether thorough, satisfactory and up-to-date throughout. Perhaps we expected the impossible; at any rate, we certainly hoped over-much.

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Testing for Metallurgical Processes. By **James A. Barr.** 216 pages. 22 illus. Price, $2. San Francisco: Mining & Scientific Press.

A commendable book for use in research and metallurgical laboratories. Very practical directions are given for the use of the instruments and appliances useful in metallurgical testing and the handling of the data obtained, so as by calculation to reach further conclusions and results, is clearly set forth. In the calibration of pyrometers the variation of the boiling points of water and sulphur with barometric pressure should have been given, and the melting point of copper should be specified as 1084° in a *reducing* atmosphere. The treatment of the subject of fluxing and slags is particularly good.

The book is recommended to students of metallurgy and to those who, by practical work, are trying to master for themselves the principles of metallurgical processes.

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Tables for the Determination of Common Rocks. By **Oliver Bowles**, M. A., instructor in geology and minerology, University of Minnesota. 16mo., 64 pages. Price, $0.50 net. New York: D. Van Nostrand & Company.

The author says: "Rocks can never be learned from a text-book alone." We will go further than this and say: An abridged text-book such as this, is of practically no assistance under any circumstances in *learning* rocks. If you know the rocks, the book can serve simply as an *aide de memoire*, if you do not know the rocks, it is of no assistance whatever in helping to know them. The compilation is occasionally somewhat careless; for instance, the red, bluish and green plagioclase feldspars are left out of the list of rock-forming minerals.

The publishers should be scored for putting 84 pages of advertisements in a book having only 64 pages of text. It is a rank imposition on the man who is supposed to carry the book around with him.

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Elementary Crystallography. By **W. S. Bayley.** 239 pages, 311 illus. Price, $2. New York: McGraw-Hill Book Company.

This book, Part I of a general system of mineralogy, and published as a separate volume, is designed to introduce the reader to the subject of crystallography. Only a slight knowledge of physics, chemistry, and mathematics is required to comprehend the concise and generally well-written and profusely illustrated text.

But to the reviewer the work lacks one essential element in the presentation that unfortunately is missing from all modern books on this subject—that is, an adaptation of the usual academic presentation to practical use. Ninety-nine per cent of all students of crystallography intend to apply it only to the recognition of mineral forms, and there drop it. The presentation of the crystal forms as geometrically symmetrical solids does not assist in this practical application of the sub-

ject, and the use of the ordinary wooden models but little more. What is wanted is a work that gives more detail to the description of the non-geometrically symmetrical crystal forms as they actually exist—that is, the presentation to the student of the forms having missing planes and the crystals of unequal growth, in fact, of the crystal forms as they are usually found and not as the crystallographer thinks they should be. For instance, the author defines a cube as a solid consisting of six planes, each a *square* at right angles to the isometric axes. The actual number of square planes on the average crystallographic cube is rarely more than two, and later on illustrations are given of two cubes with but two square planes each. The author is not consistent in this regard.

The use of Naumann's symbols for indicating crystallographic forms is rather obsolete, all authoritative works on mineralogy and chemical crystallography using Miller's indices. Similarly, the terms used to name the various forms should have their more common synonyms given, as found in such standard works as Dana.

The chapters of Physical and Chemical Crystallography are welcome additions to an elementary book and are in keeping with the general plan of the more extensive chapters on Geometric Crystallography. The introduction of these chapters alone makes the work a notable improvement over any previous book of the same grade.

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The M. C. C. Periodic Chart of the Elements, with introduction. Price, folded and bound, 8s. 6d.; roll form, 12s. 6d. London, W. C. (333 Grays Inn Road): The Metals Composition Company.

This is a chart 40 in. wide by 34 in. high, having on it 120 spaces 2 in. x 4 in., representing a periodic arrangement of the elements. For each element is printed in the space its principal chemical and physical properties. In the vacant spaces are printed condensed physical and chemical tables and other miscellaneous information. Ten groups of elements have been used, instead of the familiar eight, the distinction arising from making three groups of Mendeleeff's group VIII. The chart is clear, and will be of considerable interest in the chemical lecture-room or laboratory, hung up on the wall for casual reference.

Part I of the instruction is a straightforward and sensible explanation of the principles of the periodic classification of the elements; part II is a dissertation on "the periodic law and its relation to speculative thought." The latter part is distinctly out of place in connection with the educational aspects of the chart, for it deals with such philosophical abstractions as "Life: a Mystery" and "Universal Ultimate Relationship." It shows a wide range of solid reading and thinking on the part of the author, but the proper place for its presentation would be to read it before some philosophical society and see what would be the judgment of his peers upon his speculations.

Altogether, and considering by whom it is printed, it has such a tincture of the commercial and the alchemistical as "queers" it when we try to take it most seriously. Nevertheless, the chart may be found quite convenient and useful.

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The Modern Manufacture of Portland Cement. By **Percy C. H. West.** Vol. 1, Machinery and Kilns. Price, $4 net. New York: The McGraw-Hill Book Company.

The manufacture of cement has increased in the past decade by leaps and bounds in this country, and Germany especially. The evolution from the old hand-raked reverberatory to the shaft kiln has been concomitant with the increasing uses of cement in the building trades. While text-books on the testing of cement and on concrete construction, especially reinforced concrete, are numerous enough in recent technical English publications, yet the practical and commercial side of making cement has been neglected. This we opine is due to the rapid flux which cement practice has undergone in the past ten years.

Accordingly it will be seen that this book of Mr. West is written at a moment when changes have become less and less noticeable and a standard of practice, or rather an approximate to a standard, has been obtained. Though the book in our opinion does not give due credit to the work of earlier American pioneers and to recent American pioneers as Edison, with the long rotary kiln and his pneumatic system of separation of the fines, yet the very fact that it does give prominence to the heavy and substantial machinery of the English cement works —a substantiality that often approaches mechanical corpulency —makes the book valuable to the American engineer, whose tendency is rather to a maximum of efficiency for a short period.

In the succeeding volume we hope that the application of physical chemistry to the practical problems of cement making and the question of thermal efficiency of the various systems, for each of which this journal is ever a constant advocate, will be discussed in the fullest proportions.

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Outlines of Chemistry. A text-book for college students. By **Louis Kahlenberg,** Ph. D., professor of chemistry in the University of Wisconsin. 548 pages, 155 illus. Price, $2.60 net. New York: The Macmillan Company.

Why do our students spend so many years studying a foreign language and know so little of it at the end? Because they are troubled with the rules of good grammar before they can read or speak the language. Why do so many students know so little about real chemistry, although they "have been through" a chemical course? Because they have been pestered with theories and speculations before they knew the simple fundamental facts of chemistry, and the result has been to kill their proper interest in chemistry as an *observational* science.

We think Professor Kahlenberg exactly right when he puts only experiment and observation into the first five chapters of his splendid book, excluding from them even all reference to any theories whatever. This is the way to provide the student with facts to think about, to stimulate in a healthy way his curiosity, which gradually develops, as to what explanations there can be for these facts. This is the normal and natural way to acquire knowledge of theory; not to have it pumped into the observer just as he is acquiring his fundamental facts or even before he gets them. It is only after a student has tried to imagine for himself some rational explanations for the facts he has observed that he can properly understand, appreciate, and assimilate theory presented to him by others.

With these principles to start with and building thereon a system of instruction in elementary chemistry which is marked throughout by its appeal to the student's observing and reasoning powers (as to oppose a system of cramming into him a flood of facts mixed with cut-and-dried theoretical explanations) the experienced author and teacher has given us a wonderful book. There is only one other which we can compare it with—Mendeleeff's classical "Principles of Chemistry," and it is better than that for the purpose it is intended to serve—a one-year course in chemistry. Mendeleeff's work would need at least two years to work through, and is not so well arranged for the very beginner in the science.

The long experience of Professor Kahlenberg shows itself on almost every page of his book. It is clear, tactful, even sympathetic in places, in dealing with matters not easy to comprehend. The facts are carefully systematized, theories are skilfully introduced and presented in a larger-minded way, and the total result is a "work of art" in instruction in elementary chemistry which has probably no equal and certainly no superior.

We predict that our American educators will quickly appreciate the merits of this book and that it will have a phenomenal success.